NOTES ON LEAD AND COPPER
SMELTING AND COPPER CONVERTING

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LEAD AND COPPER SMELTING

AND COPPER CONVERTING

REVISED WITH ACCOUNTS OF TWELVE YEARS EXPERIMENT AND DEVELOPMENT

BY

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PREFACE TO THE FIRST EDITION

THIS book is precisely what is indicated by its title — a series of notes on the practical work in lead and copper smelting, including the converting of copper matte. It is by no means a treatise or an attempt at a treatise. No effort has been made to trace in order all the steps in beneficiating ores by smelting from the crude material to the marketable product. This has already been done ably by other writers. It has seemed, however, that the experience gained in the every-day operation of three large works, extending over a period of ten years, might be useful to others who are engaged in similar work. Progress in any art is helped by an interchange of ideas. Hence these notes are offered.

Acknowledgment is due Mr. Wm. Braden for the reproduction of drawings of the settlers used at the Arkansas Valley Smelting Works, Leadville, Colo., from his paper in the *Transactions of the American Institute of Mining Engineers*, Vol. XXVI; and to L. S. Austin for the illustrations of the matte-pots and slag-trucks employed at the Omaha & Grant Smelting Works, Denver, Colo., which are taken from his paper on the separation and disposal of slag in *The Engineering and Mining Journal* of November 23, 1895; also to Julius A. Dyblie and John Bendixen for valuable services in preparation of plans and drawings.

PREFACE TO THE REVISED EDITION

In revising the text of this series of notes on Lead and Copper Smelting and Copper Converting I shall make but few changes in the original, even when in the light of later experience I should have stated the matter differently. A postscript will be written to each chapter, giving in as condensed form as possible the progress in the science, since these notes were written in 1896. The readers of metallurgical works have given a fair portion of their support to the preceding editions, and I trust that they may find that such contradictions as occur in the newer portions show a healthy search for truth.

Philadelphia, *March*, 1908.

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COPPER MATTE SMELTING

THE MATTING of ores without the presence of copper to serve as a carrier to collect the silver has been attempted on many occasions, but has only resulted in failure, the losses being too large under ordinary conditions to admit of treatment in this way. A small percentage of copper can be used with success, but if the resulting matte runs below 5 per cent. copper, it is very doubtful if the process can be made successful. In cases where a large tonnage of ore is to be treated with a small amount of copper, it would be advisable to resmelt a part of the matte produced along with the charge to supply the copper needed. In cases where the ores are sulphides, without the presence of either copper or lead to act as a carrier, better results can be obtained by leaching in localities where the conditions will not admit of marketing the ore to custom smelters. The reasons are that the tonnage of matte produced is not easily transported, and when sold has to pay treatment charges as well as allow for losses in subsequent treatment. If it is possible by pyritic smelting of the ore and matte to save enough copper to overcome these objections, then it becomes a question of costs to determine which is the most economical, and each particular case is an independent proposition and should be treated accordingly.

Changes of ore supply occur, and it may be necessary to convert a lead-smelting plant into a mixed one for lead and copper, or *vice versa.*

In 1890 at the works of the Arkansas Valley Smelting Co., in Leadville, copper ores of sufficient quantity to justify separate treatment were received, and accordingly two of the old lead furnaces were changed to copper furnaces by the very simple process of filling up the lead crucible and placing an overflow pot under the slag-tap, which pot was later replaced by a forehearth of the type shown in Figs. 1 to 4.

The running of these furnaces was, at the time, a novel feature

in Leadville practice, as prior to that time all the blast-furnace work had been lead smelting and no strictly copper smelting had been done. The copper ores were mainly from the Maid of Erin, with a considerable quantity of iron sulphides from the Mahala and Wolftone mines.

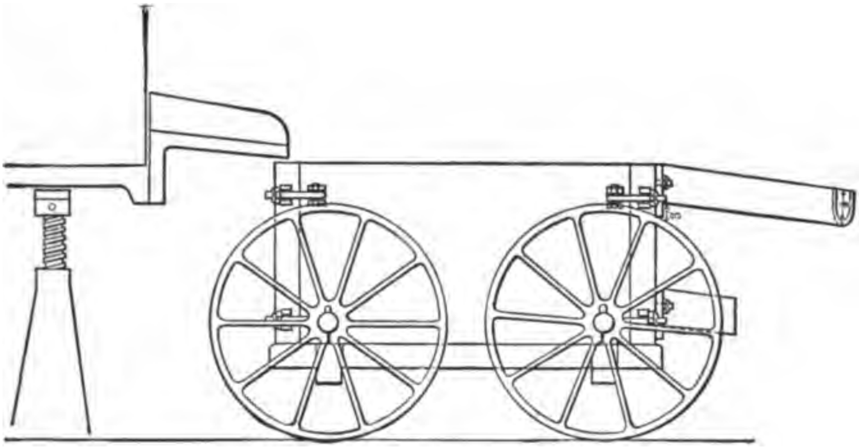
The greater portion of the ores that could be used on the charge were sulphides of such a character as to require crushing and roasting, and, as a consequence, the charge was about 70 per cent. calcines, including roasted matte from the lead furnaces, known as furnace or lead matte, which, however, contained about 7 per cent. copper, the result of concentration of small amounts of copper from the ores fed to the lead furnaces. This material was consequently fine enough to make an excessive quantity of flue dust and have a serious effect on the running of the furnace.

It might appear to the inexperienced that the size of material to be smelted would have no great effect on the cost of treatment, losses and tonnage of a furnace, but if any such persons should have to contend with the complications arising from an exceedingly fine charge, they would soon come to the belief that there is a very close relation between the size of the charge material, within certain limits, and the tonnage that can be put through.

There was a twofold object in running the copper furnace; first, to treat separately the lead ores and such ores as contained no lead and some copper, and, second, to desilverize an accumulation of foul slag of long standing which previous administrations had left on hand. This slag was in part the bottoms of pots, or that portion of the slag next to the layer of matte, and of old make, and the other part was the shells resulting from the Devereaux pots in use on the lead furnace, of which six were in blast most of the time.

It is due to state by way of explanation that lead was rather scarce in all the Colorado smelters from 1889 to 1891, and, as a consequence, the lead furnaces were running on a short allowance of lead, 8 per cent. to 10 per cent. on the charge, with a long allowance of zinc and magnesia, and the conditions were such that the slag assays were seldom below 1.5 ounce Ag and more frequently 2 and 3 ounces Ag per ton. As a consequence of the high character of bullion — 300 to 400 ounces — and low percentage of lead this was to be expected, and to remedy it as far

as possible and to prevent an excessive silver loss the greater portion of the shells was fed into the copper furnaces. If it had not been for this addition of coarse, fusible material to the charge, the furnaces would have been even more difficult to manage than they were. Being only 36×84 inches, the furnace was not large enough to put through a tonnage that would keep a constant flow open, and, as a consequence, intermittent tapping, the same as on lead furnaces, was the practice. The slag composition attempted was 34 SiO_2, 33 FeO, 20 CaO, but owing to the feeding of so much slag of unknown and varying composition



Scale ¼ inch = 1 foot.
FIG. 1.—Furnace and Forehearth No. 1.
Arkansas Valley Smelting Works, Leadville, Colo.

the analysis of the resulting slag showed up somewhat incorrectly, owing to the presence of some lead, and did not permit of the formation of as clean slags as can be made when the charge contains no lead. It is a well-known fact that in matte smelting lead is scorified, and, owing to its strong affinity for silver, will carry it into the slag, whereas if only copper and silver are present the slags will be much cleaner.

The furnaces were arranged with a double tap in front, and every time the overflow pot was changed the lower tap was opened and the accumulation of matte in the furnace drawn off. The amount thus obtained would depend very much on the condition of the furnace, and ran from one to six or eight pots of clean matte before the slag would make its appearance. The condi-

tion of the furnace was mainly dependent on the amount of matte produced by the particular charge the furnace was running on.

In cases of reconcentration before shipment, when the matte

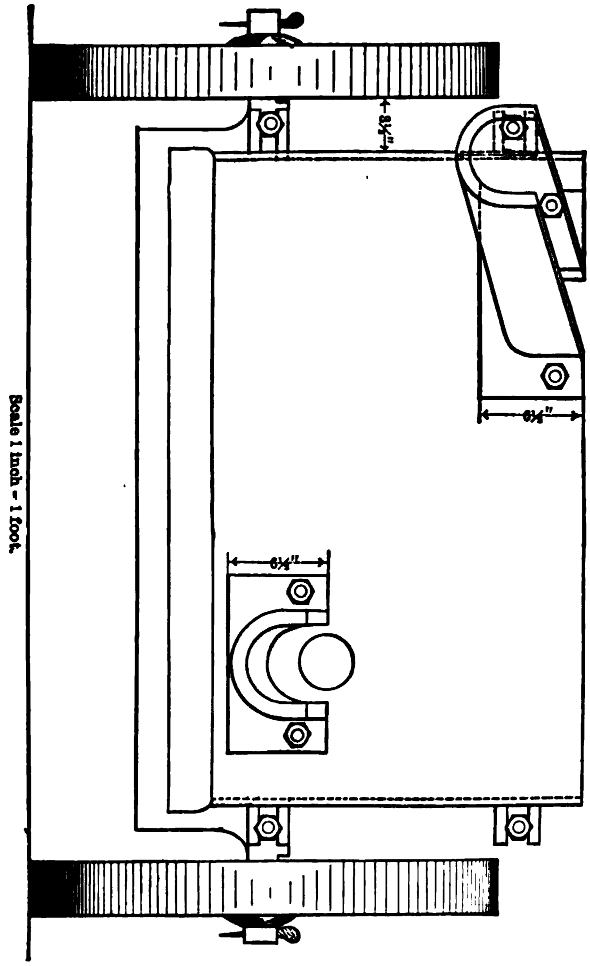


FIG. 2.—Furnace and Forehearth No. 1.
Arkansas Valley Smelting Works, Leadville, Colo.; front view.

production was very heavy, perhaps 40 per cent. of the charge, the furnace would cut out all accumulations of crust in the shaft and below the slag-tap and get into excellent condition. On the same composition of slag and percentage of fuel, with the

regular charge on, where the matte production was necessarily small to effect a concentration of ten into one or more, the zone of fusion would travel up and the bottom of the furnace become crusted to such an extent that the crucible would hold not more than one pot of matte between the slag and matte-taps.

As a consequence of small matte production and high concentration, it frequently happened that the matte-tap could not be opened until a charge was put on the furnace that would produce more matte, either by making lower grade or by feeding back some of the matte already produced.

As a general thing less than 15 per cent. matte production

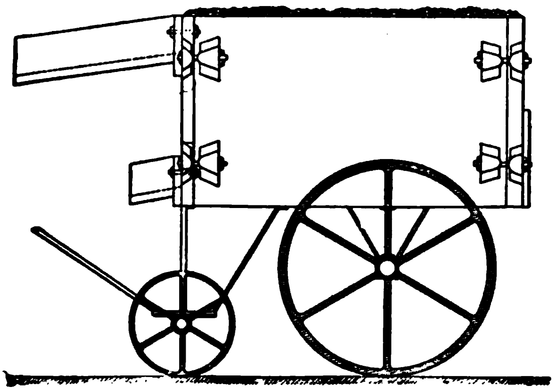


FIG. 3.—Forehearth, Furnaces Nos. 2 and 3.
Arkansas Valley Smelting Works, Leadville, Colo.

in a strictly matting furnace will result in the tuyeres becoming hard and black and the rising of the zone of fusion to the top of the charge. This, of course, results in the fuel being consumed before it should, and the consequent loss of the heat escaping up the stack, as well as the inevitable loss of silver that will follow smelting in a furnace hot on top. Many furnaces have been, and doubtless are now, run on less matte production; but to run steadily at its maximum capacity is what is expected of a well-behaved furnace, and in order to do this it is essential to have the matte production in excess of rather than below 15 per cent. Likewise the production of matte in a lead furnace has a very marked influence on the minimum amount of lead that can be used on the charge. For example, it might be quite possible to run successfully on as low as 7 or 8 per cent. of lead, reinforced

by a heavy production of matte, and altogether unprofitable to attempt it without the matte. Unless the corrosive effect of something beside slag is acting in the furnace the cold blast will chill the slag, resulting in crusts and the elevating of the zone of fusion. With the zone of fusion high above the tuyeres in a lead furnace the losses by volatilization become higher until the maximum is reached, when the entire amount is lost.

To return to the subject in hand, the handling of small furnaces with intermittent slag-tap and with matte production as low as 10 per cent. or lower, it must be said that it requires

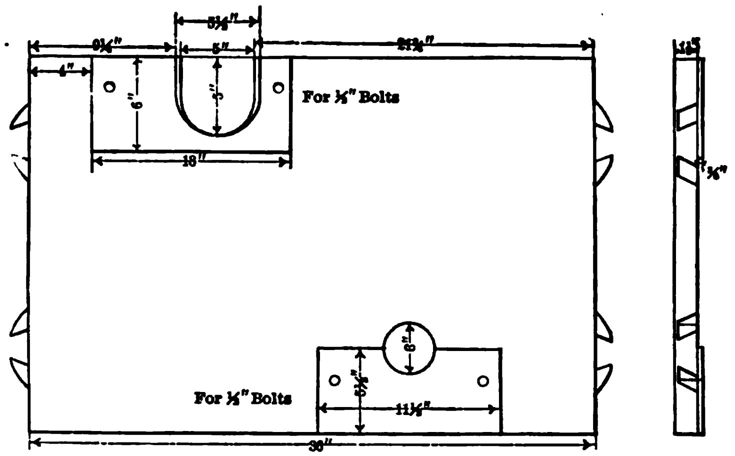


FIG. 4.—Forehearth, Furnaces Nos. 2 and 3. Arkansas Valley Smelting Works, Leadville, Colo.

very careful handling on the part of the furnace-men, even allowing that the slags are of the best possible composition. To keep the zone of fusion down at the tuyeres is the essential condition of success, and to attain this is not entirely in the hands of the man who figures the charges. The charge as it comes to the furnace may be either too fine or too coarse to obtain the best results, and while in the first case nothing but the coarsest of coke should be used, in the latter the use of a sledge hammer on the large pieces will materially improve matters.

The proper placement of the charge in the furnace is a matter of the greatest importance, and it is hard to decide where the more experience is required, on the feed floor or in handling the furnace below.

In a case where the furnace temporarily becomes cold, as is likely to happen through the variation in matte production, it may become advisable to increase the fuel or put back a charge of straight matte in order to repair the threatened bad effect, and if the feeder does not know his business, or if the furnaceman does not properly attend to his duties, the result may be slag in the tuyeres, a disagreeable experience with sledge hammers for a time, and what is known as muscular metallurgy.

When a furnace has from any cause, be it fine ore, a bad slag, small matte production, or any other fault, become dark at the tuyeres and hot on top, to know the quickest way in which to remedy the difficulty is what is required of the man in charge. If it is a copper furnace, the easiest way is by putting on matte charges until the tuyeres get bright and the crusts or accretions have disappeared sufficiently to warrant the matte charges being taken off or reduced in number by putting on one matte charge to one, two, or three, or more, of ore charges. It frequently happens at shift change, especially early in the morning when the night men are tired and are doing their work in a somewhat lazy and careless fashion, that the furnace is allowed to run down and get very hot on top. At such time the operation of the furnace assumes the character of pyritic smelting, the oxidation of sulphur being much greater than when the furnace is fed at its proper level, with the result that the matte production in proportion to the charge falls off rapidly and consequently increases in grade of copper contents. The excessive loss of heat up the stack results in the furnace running cold, and when suddenly filled up again the consequence is bad running and probable freeze-up.

It is not the purpose of the writer to refer only to ideal conditions where everybody connected with the furnace knows his business and attends to it properly, but to take from the experience of the past that which may be valuable to others in the future. Too much blast will result in driving the fire to the top of the charge, and, providing the furnace will stand it, a reduction of the blast will assist the matte charges in bringing the zone of fusion down. On the other hand, tonnage must be maintained, and in order to make a furnace run on a good charge above 100 tons per day it is necessary to drive the wind into it at a lively rate, and a No. 7 Roots blower will have to make 140 revolutions

per minute to furnish blast enough. This amount of blast would be all right for a copper furnace 42×120 inches at the tuyeres, but would be too much for a smaller furnace on the same charge, and entirely too much for a lead furnace of the same size.

It is natural to discuss at this point what is the approximate amount of air to blow in, in order to obtain the best results in copper and lead smelting. Taking the listed displacement of a Roots No. 7 blower at 65 cubic feet per revolution, and allowing 140 revolutions per minute for a copper furnace 42×120 inches at the tuyere, we get 9100 cubic feet per minute for a furnace with cross-section of 35 square feet, or 26 cubic feet of blast to each square foot of furnace section. Under ordinary conditions 70 revolutions per minute would be about the proper amount for a lead furnace of the same size to run on, and would give 13 cubic feet per square foot of furnace section.

These figures are of necessity only for normal conditions of charge and tonnage, and if conditions arise for greater or less tonnage or to reduce the loss by volatilization, the amount of blast supplied has to be regulated to suit those conditions.

Also, the blowers may, through wear, become leaky to such an extent that they will not deliver the calculated amount of air to the revolution, and, as a consequence, would have to be run faster. I have never personally tested my blowers to find out how much they would deliver per revolution and at varying pressure, except at Anaconda, where a No. 4 Roots was run with a closed outlet at 20 revolutions per minute and gave a pressure of four pounds to the square inch in the blast-pipe, which would mean that if run against a pressure of four pounds it would deliver no air at all, or, in other words, the leakage at four pounds was 100 per cent.

With Baker blowers the leakage is much higher, and I do not believe that they could be made to generate a pressure of more than two pounds before the leakage would be 100 per cent. This would indicate what experience is slowly teaching all blast-furnacemen — that the efficiency of Baker blowers, or blowers of that type, is much less at any pressure than blowers of the Roots type. When Leadville was a new camp and when the writer received his first baptism of fire in the smelting business, there was not a Roots blower in any of the smelters that were in operation, and this tendency to oppose change has been

maintained to a great degree even in the construction of new plants. It is an easily demonstrated fact that if the efficiency is lower the power required to deliver the blast to the furnaces must necessarily be larger than would be the case if blowers of the Roots type were used. As there are now two competing firms building this class of positive blowers, the writer cannot be accused of unjust prejudice.

The number of moving parts and the closer contact required for a more positive delivery of the air are the reasons for the greater efficiency and consequent lower consumption of power. The difference in power consumption will amount to fully 50 per cent. in favor of the Roots type, and regardless of first cost would soon pay for entirely new blower plants for many works now in operation. The listed speed for a No. 7½ Baker is only half that of a No. 7 Roots of the same displacement per revolution, so that on a basis of cost alone one Roots type is worth two Bakers, and on a basis of operation the ratio is two to one in favor of the Roots type, so that finally it would appear that the relative values of the two types is about four to one in favor of the Roots.

The type of furnace has also much to do with the amount of blast required for successful running. If, for instance, it is a constant discharge, the flow of slag must be rapid enough to keep the slag-spout open, and this cannot be done on much less than 60 tons of furnace charge per day. If the furnace is small and the character of charge is such as not to admit of rapid smelting, to run an intermittent tap with a moderate blast may be better than to try forcing the tonnage with high blast at the risk of driving the fire up and chilling the furnace at the tuyeres.

These were the conditions at Leadville. The ore was fine, the matte production small, and the tonnage that could be handled not enough to keep a constant discharge open. A forehearth of the Herreshoff type with water-jacketed sides was constructed, but when the furnace was started it was found that the cooling effect of the water was too great, and the entire contents of the hearth soon became solid. The hearth was changed and the water-jackets allowed to remain empty with better results, but the neck connecting the hearth with the furnace would become closed and slag would be forced out of the tuyeres after about

twelve hours' run, and it was necessary to change the hearth at least that often.

The hearth was then constructed of cast plates with a pipe cast around the hole which connected with the furnace, and a similar plate with water circulating around the hole in the furnace front. This construction worked satisfactorily as long as the matte production was small, but if as much as 20 per cent. matte was produced it would cut out and discharge the contents of the furnace on the floor. This type of furnace was not adapted to steady running under varying conditions, since the changing of the hearth necessitated a shut-down which was unavoidable and frequently of an hour's duration.

After running some time under the condition stated, with a matte production of about 10 per cent., varying in grade from 15 to 40 per cent. copper, and in silver from 150 to 600 ounces, the experiment was tried of mixing in a small percentage of speiss with the ore before roasting. At first the quantity used was one-fifth of the calciner charges, and as no bad effect was observed on the calciners or blast furnace the proportion was increased until one-half of the calciner charges was arsenical speiss that had been produced several years before and was regarded as a waste product.

This speiss contained an average of 18 ounces Ag, one-tenth ounce Au, 2 to 5 per cent. Pb, 15 to 20 As, about the same amount of sulphur, and the remainder iron. It had been made when the ores of the camp were chiefly of carbonate or oxidized character, and was a great annoyance at the time of its production, and had entailed a heavy loss of silver and gold in smelting. The accumulation of ten years' operations was stacked up in one corner of the slag-dump in the hope that by some undiscovered process it might some day be treated and the silver and gold recovered. There were about 3000 tons of it at this time on the dumps of the Arkansas Valley works, representing a large sum of money in gold and silver as well as the value of the iron excess as a flux for silica. At first this experiment did not promise to turn out well, as only the finest portions of the old pile could be used, and it was thought that when they were consumed the golden eggs would be gone. It will be understood that the crushing of speiss in lumps, varying in weight from 20 to 80 pounds and of a hemispherical shape, is attended with great risk of break-

age to the machinery, and especially when there is some metallic lead to be found associated with it. The speiss was therefore thoroughly culled before sending it to the crushing mill, and all large pieces broken into two or more pieces, by sledging, to avoid as far as possible the risk of getting lead into the crusher.

But in spite of these precautions the machinery was frequently stalled, belts were thrown off, and both belts and bolts broken, and finally the frame of a 9×15 Blake's crusher was split down through the jaws. The shaft was also badly bent and had to be straightened and rebabbitted in the bearing where the great strain had squeezed out the babbitt. But by this time rapid strides were being made in the consumption of the speiss pile that had been an eyesore for so many years, and after being encouraged by a 5 per cent. silver gain in one month's run, over and above the silver charged in the smelting returns, it was seen that the heavy repairs to several crushers could be afforded.

There was no attempt to keep the records of the copper furnaces separate from the lead smelting, since the copper smelting was carried on with a view of desilverizing dirty lead slags in combination with straight matting for direct profit; so that the 5 per cent. gain represented a total of 15,000 ounces on a charge of 300,000 ounces, and also no loss on the silver charged, which, under favorable conditions, would have amounted to about 9000 ounces, so that at a low estimate the gain was about 24,000 ounces. The contents of the speiss were not charged to the furnace, and so long as the smelting returns continued to show in that unusual and satisfactory fashion it was deemed advisable to go on utilizing it.

The crushing continued day and night; slowly, to be sure, but fast enough to use the stock on hand in about five months — all too soon for the writer, for by that time we had become speiss hungry and the dump was thoroughly explored by tunnels in the hope of finding more buried treasure.

Next in importance to the crushing came the roasting of the crushed product. This was carried on in reverberatory calciners of the standard type, with hearths 14 feet wide by 70 feet long, the speiss being mixed with crushed sulphides of copper and iron in the proportion best suited to driving off the maximum amount of arsenic and sulphur without becoming too fusible. It was found that if the speiss constituted too large a portion of

the roaster charge it was likely to become plastic as it came near the fire-box, and in that condition would stick to the hearth, as well as to the paddles and rabbles, and would naturally refuse to part with its sulphur and arsenic.

After experimenting with different roaster charges it was finally found best to use one-third speiss and two-thirds ore as giving the most satisfactory results, and this charge was continued until the stock was used up. The ore that was roasted with the speiss contained about 27 per cent. SiO_2, 25 per cent. Fe, 30 per cent. S, 7 per cent. Cu, and was admirably suited to the purpose, as the silica rendered the roaster charges more infusible than they would have been otherwise, and the copper furnished the carrier for extracting the silver in the blast-furnace. The roasters were charged four times each shift with 3000 pounds of the mixture.

The charge was worked the same as if it had been all ore, rabbled every twenty minutes and moved forward four times each shift of twelve hours. With five roasters running, each one handling twelve tons per day, we roasted twenty tons of speiss and forty tons of ore down to about 5 per cent. sulphur. The arsenic was partially driven off, about one-third the original amount remaining in the roasted product, which I am inclined to believe was oxidized to a considerable extent. The charges were drawn from the furnaces into slag-pots and dumped onto a cooling floor. In order not to alarm the men unnecessarily and at the same time to avoid danger of poisoning, the proportion of speiss to ore on the charge was increased gradually from 10 per cent. at first, to 15 per cent., 20 per cent., 30 per cent., and 40 per cent. of the charge. But with so much as 40 per cent. or 50 per cent. the fumes, when drawing the charges, became dangerous, which, with the disadvantage of the plastic condition referred to previously, made it both necessary and desirable to decrease the amount of speiss. Some of the men were badly poisoned about the nostrils, and, in fact, one was dangerously ill from the effects, but on the assurance of the company's physician that this could be guarded against, the work was kept up. A preparation of hydrated oxide of iron was put up as a salve and furnished to the men to put into their nostrils, and whether this preventive was the cause or not, we did not have any further serious trouble.

The calciners and the calcined material, after cooling, were covered by a white sublimate of arsenious acid, which had the appearance of frost on a cold morning. After cooling from twelve to twenty-four hours the roasted material was loaded on cars and sent to the blast-furnace, where it was smelted with silicious copper ores from the Sedalia mine, Colorado, or from the Eureka Hill in Utah, together with the addition of enough raw sulphides from the Leadville mines to keep the matte production at such a ratio that the furnaces would run as regular as possible. As before stated, these furnaces were originally lead furnaces, with the crucible filled with brickwork and silicious lining.

The jackets were of the ordinary cast-iron type in use in all lead smelters and of the ordinary hight, i.e., about four feet six inches. This hight of jacket does very well for a lead furnace where the blast is not high enough to raise the zone of fusion, but in the smelting of these ores there was great trouble from burning out of the brickwork over the jackets. The furnaces would treat about 40 tons of ore per day, on a 16 per cent. fuel consumption, and, in addition, 20 tons of slag. It would naturally be expected that smelting with so much speiss on the charge, a considerable quantity of speiss would be produced and would separate from the resulting matte. But such was not the case. When the furnaces were tapped it would frequently spark in the way which is characteristic of speiss, but after cooling there would be no line of separation in the pots, and upon being crushed and roasted and resmelted the product was a matte of very clean appearance with 40 to 50 per cent. copper, the arsenic contents of which did not exceed 5 per cent.

Later Experience and Notes

In looking over the matter in the preceding chapter, after twelve years of further experience in the smelting of all classes of ores, including the smelting of the copper-nickel ores of the Sudbury district in Canada, the writer sees that the necessity for a greater amount of blast, more than any other condition, marks the difference between the practice of that time and the present.

The blowers used in the early practice were so inefficient that they did not deliver the amount of air that they were sup-

posed to deliver to the furnace, and were but little better than fans. Furnaces had not been built large enough to be comparable to those at present in use, nor was sufficient ore to have supplied them, then developed.

In some cases where the ore supply had increased so as to require a larger smelting plant, the enlargement was by building more small furnaces, instead of fewer large ones.

In 1880 the plant of the Grant Smelting Company in Leadville consisted of about a dozen water-jacket furnaces, the entire product of which could be handled by one or two modern furnaces. I have been told that at Ely, Vermont, at about the same date, they had sixteen furnaces, each 36 inches in diameter, in one line. The feeders made up the charges in baskets, carried them to the furnace and mounted a ladder to throw them in.

From such conditions to the one hundred and forty foot blast-furnace at the Anaconda plant of the Amalgamated Copper Co. is quite a jump, but it is characteristic of the times, and if there were conditions of ore supply to justify it, I suppose we would have still larger blast-furnaces.

The difference between lead smelting and copper smelting that has to be kept constantly in sight is that the former is altogether a process of reduction, while the latter, in nearly every case, is a process of oxidation. The few cases of copper smelting in which the supply of sulphur is insufficient or has to be economized are to be found in new camps where the mining is not of sufficient depth to have reached the sulphide zone, or where the ores are altered lime impregnations with only a small amount of sulphides.

Of the latter class the British Columbia mines of the Boundary district are the most important example. These ores have enough sulphur to collect the copper into a 30 per cent. matte if they are crushed fine enough and fed into the furnace so that the charge will be highest along the wall and lowest in the center. If, however, the ore is crushed so coarse as to leave large lumps that make a loose porous charge, favorable to oxidation, and is fed into the furnace so that the charge is lowest along the sides and highest in the center, then the sulphur will burn off to such extent that the grade of the matte may be 50 per cent. and the slags contain $\frac{50}{100}$ Cu. If the matte is kept down to 30 per cent. the furnace will run much faster and the slags be much cleaner.

Both these conditions can be accomplished by finer crushing and by dumping the charges against the walls instead of into the middle of the furnace.

The principle involved in feeding in the two cases, is that when pieces of ore and coke of mixed sizes are dumped together on a pile the larger pieces roll to the outside of the pile. In case all the charge is dumped into the furnace from the side, in such a manner that it falls into the center, the large pieces roll to the sides and cause a line of least resistance to the passage of the blast along the walls. This line of least resistance causes overfire and oxidation; and that is the condition desired where there is a surplus of sulphur, but is just the opposite of what is desired where sulphur is scarce, or where reduction is required, as in lead smelting. Pyritic smelting is only possible where the conditions of ore supply render it necessary to waste a large amount of sulphur, and render useful in the furnace the heat developed by the oxidation of the sulphur and iron.

The only real smelting without carbonaceous fuel is done in the converter, and between that and smelting with 15 per cent. coke on the charge there are all possible variations, depending upon the nature of the ore supply. If the ore contains a large excess of sulphur and iron, together with a small amount of copper, then by giving it the requisite amount of silica, blowing a large volume of blast through the furnace and only using the minimum amount of coke, a concentration of the values takes place into a matte of ten to fifteen per cent. of the weight of ore smelted.

If the ore contains more gangue and less excess of iron and sulphur it will require less silica and more fuel, and the operation resembles less the ideal conditions in the converter, where the heat developed by the oxidation of the sulphur and iron furnishes all the heat for the smelting of the charge and the silicious lining. As a consequence of the low blast-pressure that could be developed by the type of blowers in use at the smelters up to 1890, the smelting column of all furnaces was necessarily short. With the improvements in blower construction it became possible to get more blast-pressure. When this higher blast-pressure was applied to the low smelting columns of the lead furnaces it caused overfire and higher lead losses, and in some cases caused a prejudice against the improved blowers.

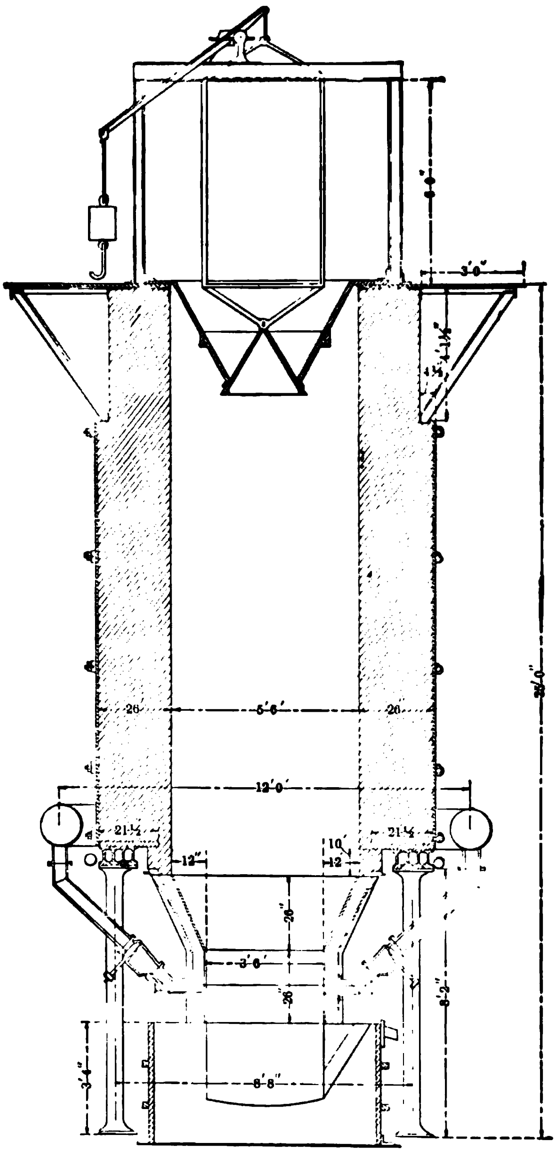


FIG. 5.

The fault was with the furnaces, and the remedy was greater depth from charge floor to tuyeres. During the reconstruction period many costly experiments were made, among them the

plant at Aguas Calientes, Mex., mentioned in Chapter VIII, was the most disastrous that came to my notice.

In the light of later experience I am convinced that the cause of the mysterious behavior of the lead furnaces there, was the bell-and-hopper feed, and not the depth of the furnaces. The bell-and-hopper feed consisted of a cast-iron furnace top with two funnels, in each funnel a cone being held up against the lower end by a wire rope and counter weights. When a charge was dumped into either funnel the cone was lowered by raising the counter weights and the charge slid into the furnace in the same manner in which iron blast-furnaces are fed. This device might have worked satisfactorily on a round furnace where one funnel and cone would have thrown all the charge against the wall and forced the coarse pieces to roll to the center; but on a rectangular furnace with two such zones of distribution intersecting, it caused lines of least resistance to the passage of the blast, and this caused overfire and loss of lead. I can see no reason why a lead furnace should not have a 20- to 25-foot smelting column, except that if the reduction is too strong iron will be reduced from the slag and make a sow in the crucible. Reduction is more affected by the percentage of fuel used than by depth of charge, so that a deeper furnace should use less fuel than a shallow one, and would require a greater pressure of blast.

II

EXTRACTION OF GOLD AND SILVER FROM MATTE

THE MATTE produced at the Arkansas Valley Smelting Company's works was shipped to the refinery at Argentine, Kansas, and treated by the improved Hunt & Douglas process, or reshipped to Block & Hartman, Belleville, Ill., and there treated by a leaching process.

In the Hunt & Douglas process the matte is roasted at very low temperature, so that copper sulphate and oxide result without forming any silver sulphate. It is then leached with dilute sulphuric acid, the gold, silver, and lead remaining in the residue. The copper solution is chloridized by the addition of chloride of lime and the copper precipitated as subchloride by passing sulphurous acid through the solution. The subchloride of copper is reduced to suboxide by milk of lime, whereby chloride of calcium for further use is recovered, while the suboxide of copper has only to be reduced to ingot by a simple smelting.

The Block & Hartman process for recovery of gold is somewhat similar to the practice at Argo, Colo., if not identical with it. The matte is first roasted to convert the silver to sulphate, then it is leached out with water and precipitated on metallic copper, the gold remaining behind in the matte. Or, it is concentrated to black copper in the reverberatory furnaces, granulated, ground, roasted, and leached with salt solution by the Augustin process and the silver precipitated in the usual manner.

The residue containing the copper and gold is then concentrated in a reverberatory furnace until a small amount of copper is extracted as a copper bottom, carrying nearly all the gold. The subsequent treatment and separation of the gold from these copper bottoms has been much talked of among metallurgists, and is supposed to be a secret of such importance that Mr. Pearce, of Argo, though generally most liberal in imparting knowledge, is, for business reasons, unable to divulge it.

An experience that occurred while running the furnaces at

Leadville bears on this point, and the reader may be left to form his own conclusions about the separation of gold from copper bottoms at Argo. In concentrating the matte which, owing to the large percentage of lead slag on the charge, contained a small amount of lead, there were formed some bottoms which, while they could not be called properly copper bottoms, will answer for an illustration. These bottoms were metallic in appearance and would ring like bell metal when struck with a hammer, were quite malleable, and could not be broken. The matte, which separated from them very easily, did not contain more than 55 per cent. Cu, so that without the intermixture of lead they would not have been formed. Having them on hand it became a question what to do with them. An assay showed that the gold had almost all left the matte and gone into these bottoms. The experiment of fusing a portion in a scorifier with the addition of a pyritous ore containing no silver and a little gold was then tried. It was found that the gold continued to concentrate in the portion which remained metallic, while very little, if any, went into the matte formed by the pyrites on the surface of the fused charge.

In fact, there was a scorifying action, with sulphur as the agent of concentration, and the resulting metallic portion all the time growing richer in gold as it got smaller. This action was kept up until the greater portion of the copper had been removed, and the resulting button could be easily cupelled with lead, whereby the remaining copper was removed, and a gold button remained which could be dissolved and precipitated as fine gold. Certainly the gold can be extracted from copper bottoms in this way, and since it is a well-known fact that mill concentrates and even tailings are to be had from certain gold mines around Black Hawk, Colo., with which the excess copper could be reconverted into matte and the gold concentrated into such a small amount of copper as to admit of its being refined, it remains for the reader to decide whether it is a secret process or only one of the tricks of the trade. It is at all events only applicable to just such conditions as exist in this process of extracting the silver first and subsequently the gold, and it is difficult for any one to see how a competitor could possibly take advantage of the general knowledge of the process, especially as the electrolytic method is better and cheaper.

Later Experience: The Electrolytic Process

The electrolytic process of refining and separating copper from the precious metals has made such progress in the last twelve years that it is in entire possession of the field. All of the older processes for the recovery of gold and silver from matte have been abandoned, and as a consequence copper has largely displaced lead as a collector of these metals in the customs smelting of other ores.

One substantial reason for this is that the electrolytic refining of copper improves its qualities to such an extent as to put it on a footing of equality with lake copper. The price obtainable for electrolytic copper is enough greater than the price of casting to pay the cost of refining, so that it is almost certain to be refined electrolytically, and, therefore, might just as well act as a collectory for the precious metals in other ores.

The improvement in physical electrical qualities has caused some of the lake copper to be refined electrolytically in order to hold its trade position as the best copper that can be produced. This tells the story plainer than any argument could that the best grades of electrolytic copper are equal to, or better, than lake copper, and it tells further that prejudice costs the consumers the difference between the two prices.

III

THE CALCULATION OF FURNACE CHARGES

THE CALCULATION of a slag for the furnace is illustrated by the following approximate make-up of the charges we were using in matte smelting at the Arkansas Valley works, as described in a previous chapter, taking the analysis of the ores from memory. They may not be exact, but they are close enough for the purpose of illustrating the method ordinarily used by metallurgists for figuring blast-furnace charges.

	Weight in lbs.	Per cent. SiO_2	Pounds SiO_2	Per cent FeO	Pounds FeO	Per cent. CaO	Pounds CaO	Per cent. S	Pounds S	Per cent. Cu	Pounds Cu
Calcined ore and speiss	500	20	100	42	210	0	0	5	25	5	25
Raw sulphide ore ...	150	26	39	32	48	1	1.5	30	45	8	12
Silicious ore	150	53	80	14	21	6	9	2	3	3	4
Lime rock	200	3	6	..	..	52	104	..	..	..	..
	1000	..	225	..	279	..	114	..	73	..	41

There is no algebraic mystery or X, Y, and Z equation to be solved by higher mathematical methods than percentage, and the results are just as reliable. There is generally a great deal of mystery thrown about this portion of the metallurgist's work, as if there were a fear that if the younger members of the profession were put in possession of the combination they might soon be competitors.

The writer has no hesitation in saying that the calculation of a charge, when it has been decided what ores are to be smelted, is the simplest thing about the work. The difficulty is more often to decide what to put on, and how much matte to make, to have it run to the best advantage. There are certain things a metal-

lurgist must know; for instance, he must be able to tell approximately how much matte the charge is going to make and how much of the iron is going into it, in order to make the proper deduction from the total amount on the charge. The writer has been accustomed to have all analyses of ores, whether sulphides or oxides, determined or figured as FeO, and then, knowing from experience and from the running of the furnace about the grade of matte that certain charges will produce, he proceeds to multiply the total pounds of copper by a certain factor to get the weight of matte that the charge will produce.

In the case of the charge above cited let us assume that the matte will assay 25 per cent. Cu. He accordingly multiplies the 41 pounds of copper in the charge by four, which gives the matte production at 164 pounds. A 25 per cent. Cu matte with the amount of impurities, such as Pb and As, that would naturally go into it from ores of ordinary character, would contain the equivalent of 40 per cent. FeO, and 40 by 164 equals 65.6 pounds FeO, to be deducted from the total of 279, and leaving off the tenths gives 214 pounds FeO to go into slag. The sum total of the SiO_2, FeO, and CaO, after deducting the iron that will go into the matte, is 553 pounds, which, according to slag determinations, is generally 90 per cent. of the slag.

Accordingly 553 pounds of SiO_2, FeO, and CaO, when put into slag together with Al_2O_3 and other oxides, would represent approximately 620 pounds slag to a 1000-pound charge. These 620 pounds divided into the amounts of SiO_2, FeO, and CaO that would go into the slag would give a calculated analysis of 36.3 per cent. SiO_2, 34.5 per cent. FeO, and 18.4 per cent. CaO.

In explanation of the reason why it is assumed that the matte produced will assay 25 per cent. Cu, it may be stated that this would be the experience with ordinary running of the furnace on the charge figured, the excess of the sulphur being burned off. The grade of matte being dependent on the condition of the furnace, the depth of charge, and the amount of blast used, a knowledge of the effect of these conditions is necessary in order to make this assumption.

As regards the percentage of Fe in different grades of matte, that is a matter of local experiment depending on the amount of other impurities that the matte can absorb from the ores on the charge. For example, if the charge contained no arsenic, anti-

mony, or zinc, the iron contents of a 25 per cent. copper matte resulting from smelting them, would be higher than if these impurities were present in the ores. If present they will go partly into the matte and will displace iron.

Roughly speaking, matte is a compound of the sulphides, arsenides, and antimonides of the metals, while slag is a union of the oxides of the metals with the oxide of silicon. The characteristics of either matte or slag as regards melting point, conductivity, and specific gravity are capable of as many variations as the composition. As in the case with the alloys of the metals themselves, where by certain combinations it is possible to produce one that will melt at a much lower temperature than any of its constituents, so it is with slags and mattes, but in a less marked degree. There are certain combinations of SiO_2, FeO, and CaO that form the more fusible slags, and are adapted for certain purposes.

The science of metallurgy is the application of this knowledge to the formation of fusible compounds in order to assist in the recovery of the metals or their sulphides, arsenides, or antimonides. In the case of the charge figured above, the slag would be silicious enough to insure the driving off of a reasonable amount of sulphur and arsenic and still near enough to a neutral slag to flow freely, and in view of the grade of matte produced should be clean, or at least as low as 1 per cent. of the matte assay.

The assay of slags is a thing as much dependent upon the means provided for settling them, and the care with which they are handled, as upon their composition. Still it bears a very close relation to the assays of the matte or bullion, which are closely related to each other, subject to varying conditions. Generally speaking, the slag will assay 1 per cent. of the matte in copper work and 2 per cent. in lead work. In the latter case the matte will, as a rule, contain about one-fifth as much silver per ton as the bullion if the amount of matte is normal; but if the matte production is large, compared to lead, the relation between assays will fall lower. In a case where 400-ounce bullion was being produced on a 11 per cent. lead charge with 5 per cent. of matte, the matte would assay approximately 80 ounces, and the slags, if well settled, 1.6 to 2 ounces. But these things are so variable that hardly any rules can be given that will not have as many exceptions as applications.

The calculation of a charge for the copper furnaces at Aguas Calientes, Mexico, is here given:

	Weight in lbs.	Per cent. SiO_2	Pounds SiO_2	Per cent. FeO	Pounds FeO	Per cent. CaO	Pounds CaO	Per cent. S	Pounds S	Per cent. Cu	Pounds Cu
General mixture .	500	20	100	30	150	7	35	20	100	6	30
Mixture.........	200	20	40	36	72	2	4	33	66	0	0
Silicious ore	200	50	100	7	14	3	6	4	8	0	0
Copper ore	100	20	20	19	19	4	4	22	22	19	19
Converter slag ...	150	25	37	62	93	0	0	0	0	5	7
Lime rock.......	260	3	8	..	..	52	135	..	..	..	..
Concentrates	90	19	17	32	29	2	2	21	18	10	9
	1500	..	322	..	377	..	186	..	214	..	65

Assuming a 25 per cent. Cu matte, 65 pounds Cu $\times$ 4 = 260 pounds matte $\times$ 40 per cent. = 104 pounds FeO to be deducted for matte, leaving 273 FeO to go into the slag. 322 SiO_2, + 273 FeO, + 186 CaO = 781 $\div$ 9 $\doteq$ 870 pounds slag to the charge. Dividing 870 into 322 SiO_2 = 37 per cent.; into 273 FeO = 31.3 per cent.; into 183 CaO = 21.0 per cent.

In the figuring of charges the estimation of the SiO_2, FeO, and CaO contents to the one-tenth of pounds is a fineness of calculation that is totally wasted on the man with the shovel at the charge scales. Besides, many other conditions prevail which do not warrant the assumption that the ores are uniformly of the same composition. The bedding may not be regular, and when a new bed is opened more of the top layers go into the charges than later; and if the bed is in a square bin, as the opposite side is approached, the top layers are exhausted and the bottom layers alone will, for a time, take the place of the entire bed. In the face of such conditions, calculations that may have been perfect melt into nothingness, and it becomes necessary to be able to tell the character of the slag as it comes from the furnace and to make such corrections as are necessary, frequently without analysis.

The larger the ore mixtures the more evenly the furnaces

will run, but if the mixtures are small and it is necessary to change the charge every day, there is not sufficient time to settle down to business and get the charge corrected to just the proper point to do the best work. The weighing of the charges is a part of the operation that requires the constant attention of a careful foreman. Owing to the scattering of fine ore over the scales while shoveling in from the bins to the car or barrow, the scales should be swept clean after each charge and the accuracy of the balance frequently verified.

The character of slag to be run on is largely a matter of personal preference with different metallurgists, but in general it is determined by the local conditions regulating the cost of iron and lime.

The limits within which the experience of the writer has reached the best results are for copper blast-furnace work:

SiO_2, 30 to 38 per cent.; FeO, 30 to 40 per cent.;
CaO, 10 to 25 per cent.

But under ordinary conditions the slag that is best adapted to a constant discharge and regular run is:

36 per cent. SiO_2, 33 per cent. FeO, 21 per cent. CaO.

It may change considerably either way, and continue to run just as well, and a slight correction will bring it back to the original type.

I do not pretend to say that slags cannot be made higher in SiO_2 than 38 per cent., but they run very slowly compared to slags with less, and the coke seems to burn out faster than the charge smelts, leaving the furnace full of cold stuff, and accretions form rapidly at the tuyeres where the blast strikes the slag.

It has been stated in other works that the composition of the slag may vary without serious results within much wider limits than those stated by the writer. While this is true of furnaces where the slag is tapped periodically and can collect in the furnace and run out with a rush, it is not true of a constant flow furnace, as any such slags would solidify in the spout, and before the furnace could be opened the accumulation in it would run out from the tuyeres. At Anaconda, where the charge was largely slag from the copper converters, I have often had the analysis show 40 SiO_2, 43 FeO, 9 CaO when the matte production

was large (about 30 per cent. of charge), but in spite of this the furnace would run much slower and the spout would be difficult to manage. It was found that when the SiO_2 was near 36 per cent. and the sum of the FeO and CaO about 54 per cent., the best results were obtained. This was also the experience with the copper furnaces at Aguas Calientes, and while it might appear to be a better commercial proposition to make a more silicious slag, still, if the furnaces will not smelt enough ore on that kind of slag to supply the material to keep up the circulation in the spout and prevent its freezing, it does not pay to run on such slags.

Later Experiences: Notes

Furnaces are now built so much larger than they were at the date the above was written, and they now smelt so much more material in a given time, that the circulation in the slag spout and settler are greatly increased. The increased amount of slag and matte discharged through the spout and settler has caused trouble of another nature. The difficulties are to keep them from cutting through where formerly, with the smaller furnaces, they froze up. All this goes to show that what is true of one time and set of conditions is not true for all time and for all conditions. Slags are now made running as high as 48 per cent. SiO_2 at the works of the Granby Consolidated Co. in British Columbia, but it is done on a furnace capable of smelting 400 tons per day in place of 150 tons per day for the furnace referred to.

IV

TYPES OF FURNACES

THE TYPE of furnace has much to do with the kind of slags that can be run in it. An intermittent tap will handle slags that would not do at all for a constant flow, for the reason that in the latter case the slag would chill so quickly that the tap-hole would be choked up, causing shut-downs that in many cases would develop into freeze-ups. The best type of intermittent tap is a single slag-tap in front and matte-tap on the side at about one foot lower level. The hearth should incline forward from a foot below the level of the tuyeres at the back to two feet in front. This is a better arrangement than having the matte-tap immediately below the slag-tap in front, for the reason that the matte can be tapped at any time, while the slag is running, without removing the settler. The constant-flow furnace, on the other hand, gives better results when the matte and slag are run into a forehearth of from 5 to 10 feet in diameter and are given ample time to separate. The volume held in ordinary overflow pots is too small for complete separation. Like the deposition of sediment from water, matte and slag must come to a standstill in order to separate thoroughly.

A forehearth requires considerably more tonnage from the furnace to keep it open than the shallow crucible and overflow pot, and also, as the settler is made larger, more matte on the charge; otherwise the radiation is enough to form a thick crust of slag on the top and sides and to greatly diminish the size of the settler. The interior of the settler takes the shape of a pear with the larger end down, where the corrosive action of the matte is the greatest, while in the upper portion the slag once chilled remains there.

The handling of furnaces with internal crucible is attended with about the same amount of difficulty as those with forehearths, the difference being that in the first case, if it is impossible to tap the matte from the furnace, it can be run out into

the settler in front, until such times as the accumulation in the furnace renders the breast soft enough to drive a bar. But if the matte-tap on the forehearth is lost through failure to put in a bar immediately after tapping matte and while it is yet soft, or through chilling of the contents owing to slow running of the furnace, then all is lost; for while the furnace may be put into good condition in a few hours by matte charges, the settler being lost and too large to move, there is nothing to do but blow out the furnace and dig out the settler. This is the only bad feature about large settlers; every other point is in their favor, especially in connection with converters, as they will permit a sufficient amount of matte to accumulate for the converter charge.

The separation between matte and slag is much better in large than in small settlers, but the weight of a settler 10 feet in diameter by 4½ feet deep, filled with slag and matte, is too great to move unless extraordinary appliances are provided. In all cases after tapping matte, unless the breast is exceedingly soft, as soon as the flow has been stopped with clay, a bar should be driven in slowly until the point has just penetrated the clay and entered the crucible or hearth. This is to provide against a hard breast when the next tap is to be made, and is a very important detail to attend to in the running of the furnace. If, for instance, it is neglected, as it is likely to be, owing to the attention of the men being drawn to other things, it may mean several hours of sledging to again open it although the breast be reasonably thin.

With the matte-tap immediately below the slag-spout, as was the case in Leadville, the slag was drawn off until the blast blew out at the tap-hole. The settler was then removed as quickly as possible and the bar driven into the matte-tap, a slag-pot run under the matte-spout and the bar withdrawn by means of the ring and wedges, which are absolutely necessary. All the matte in the furnace would be tapped out without stopping the flow, except for a few seconds to change pots. The stoppage was done by means of a piece of iron shaped like a tinner's soldering-iron, about 15 inches long and 2½ inches in diameter, welded to a gas-pipe handle about 10 feet long. The heavy point would be inserted in the tap-hole and the stream interrupted until the full pot could be drawn away and an empty one substituted.

The tap-hole in front is not a good arrangement. It should be on the side of the furnace, but as near the front as possible. Its position in front at Leadville was unavoidable on account of the way in which the furnaces were crowded together. It frequently happened that after the settler had been removed and before the matte-tap could be opened, the furnace would be full of slag and the slag-tap have to be opened again to prevent it running out at the tuyeres, which it sometimes did. With the matte-tap on the side the work of tapping matte can go on at any time without interfering with the slag-tap or necessitating the removal of the settler. This point is worth mentioning, because if it is known how not to do a thing it may be of great assistance in finding out how to do it.

At San Luis Potosi, Mexico, they have a number of lead furnaces converted into matting furnaces by filling the crucible and putting in a sloping hearth of fire-brick. The crucible is about two feet deep below the tuyeres in front, and a tap-jacket is put in on the side near the corner of the furnace, high enough above the floor to admit a slag-pot under it, with slightly depressed floor space immediately around the tap. These furnaces are used for the concentration of matte made in the lead furnaces; whatever metallic lead is produced comes out with the matte and is recovered on the dump.

Running a blast-furnace in connection with a converter plant of the size of the one at Anaconda, where the sole purpose is to work up refuse material and smelt over converter slag, is a much simpler operation than smelting ores which are widely different in character and only mixed mechanically in the charge. The fact that a portion of the charge has been fused previously, no matter what the composition of the fused material may be, has a very beneficial effect on the smelting. This is probably due to the fact that the fused material melts higher up in the shaft than the unfused and has a dissolving and uniting effect on the constituents of the latter.

With a charge that was 60 per cent. converter slag, it was only necessary to add raw ore to make the matte sufficiently low to extract the copper from the slag and waste material, together with lime rock to assist in the fluxing of the surplus SiO_2. The FeO was already in the converter slag and in combination with SiO_2, so that such smelting cannot be said to require any

special mention from a scientific point of view, but merely as a matter of practice in connection with converting to recover the copper in the slag.

Later Experience with Furnaces

To recover a settler that has partially chilled it is sometimes necessary to drill out the tap-hole just as a hole would be drilled for blasting in mining. To do this it is necessary to start with a large bit and cut a large hole into the hardened material as far as possible, or as far as is necessary. Smaller bits are then used in graded sizes until a point is reached where the tapping bar can be driven through to the liquid interior. The bits of these drills should be sharpened so as to cut a hole larger than the bar of which the drill is made, to prevent the bar being stuck in the hole. The drill should be turned just as in drilling rock and no attempt made to drive until the hole is deep enough to justify it. Thermit, which is used for welding rails and castings, might be used to advantage in such cases to enlarge the tap-hole. I have heard of one case at Anaconda in which a pipe was inserted in the top of the settler and a blast of air blown in to develop heat through the partial converting of the matte in the settler. I have no doubt that this would be effective, but it would foul all the slag in the settler. It would appear that by the use of Thermit on the tap-hole a sufficient amount of heat could be developed to fuse the passageway into the liquid interior.

While copper blast-furnaces that are more than 100 feet long have been built, and are now in operation, the width has not increased much and is generally between 42 and 60 inches at the tuyeres. Lead furnaces have not been increased to such large dimensions because they require more care in handling and will not stand the helter-skelter type of feeding practiced on copper furnaces. If the charges were dumped into a lead furnace in the manner that they are dumped into the average copper furnace, it would stop producing lead in about the time required for the charge to get down to the tuyeres. The action of a properly run lead furnace is exactly the opposite of that desired for nearly all copper furnaces. The former must be handled so as to produce the greatest amount of reduction at the lowest temperature possible in order to prevent the loss of lead, while the latter are

in nearly all cases run as oxidizers, because it is desired to burn off the surplus of sulphur in the ore, over and above what is necessary to make the matte, to collect the copper and other metals.

If lead furnaces were made to be tapped on the side instead of on the end, and the lead well were placed on the opposite side, in line with the slag-tap, there does not appear to be any reason why a lead furnace should not be built as long as a copper furnace. The feeding would have to be entirely different and so arranged that the charge as dumped into the furnace would be spread and go against the walls. In this way the finest pieces would follow down the furnace walls and the coarse pieces roll to the centers of the furnace, causing the blast and gases to distribute evenly through the charge. A small portion of the charge would probably have to be fed into special places with a shovel, to prevent blow-holes and overfire, which are so fatal to good results in lead smelting. Good results can be obtained in lead smelting by proper feeding when the slag is not of the best composition, but if the feeding is wrong the results are poor, no matter what the slag composition may be. For producing the best results the feeding has more influence on a lead furnace than the composition of the slag. The pieces of ore and limestone and coke should not exceed a size that will pass a four-inch ring, and should on the other hand not contain more than 10 per cent. material that will be blown into the flue.

V

SPOUTS, SETTLERS, AND JACKETS

THE FORM of spout and tap-jacket for a constant-flow furnace is a very important matter, and is one of which no special mention has been made in any work on the subject. The spout ordinarily in use in Montana in early days was made of one-inch pipe put together as closely as possible, and in such a way as to form a channel of four and one-half feet long, semicylindrical in shape, with one end open and one partially closed. A piece of sheet iron was fastened to the curved outside, and clay, or a mixture of ground quartz and clay, was rammed in between the pipe and against the sheet iron. When in position, with the open end against the front of the furnace (see drawing of Anaconda furnace) this spout allows the slag to flow in a steady stream into the forehearth without the escape of blast, the slag stream flowing over the partially closed end and thereby trapping the blast. This spout is known as the Schumacher spout, and was in use for many years. It does good work, but the weak point about it is that the quartz or clay is eaten out by the slag and matte, and frequently the contents of the furnace flow through some defective point in the spout.

To overcome these defects and prevent runaways the writer had the coil cast into an iron spout of the shape shown in Figs. 6 and 7. It was found that fewer pipes were necessary for this kind of spout than for the old construction, and the number of coils was reduced to four.

The cast iron proved to be sufficiently cooled by the water circulating through the coils to keep the spout from being attacked by the matte, except at the tip, where the stream falls into the settler. At this point the cast-iron covering was eaten away and the pipe was soon laid bare.

If the pipe used in making the spout is not of the best quality it may soon be eaten away at this point and the spout rendered worthless. This is the most satisfactory spout that the writer

has ever used on a constant-discharge furnace, as the cooling effect of the water is reduced to a minimum and is only what is required to keep the spout from being eaten out by the matte.

The pipe used may be of any size, say from ¾ to 1¼ inches, according to the water pressure, but must be of the best quality and of extra thickness. The coil should be made very carefully and put together with malleable ells and return bands. It should

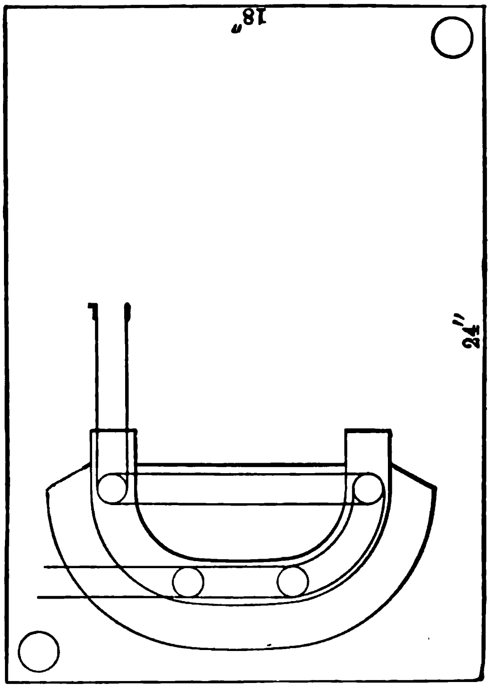


FIG. 6.—The Hixon Slag-spout; Cross-section.

then be heated to drive off all grease or oils, and while warm painted with graphite mixed in benzine. Several coats of this should be put on and allowed to dry before the coil is put into the sand mould to receive the cast-iron covering. The cast should be poured as cold as possible in order to avoid burning the pipe.

Any good foundryman should be able to make these spouts without much difficulty, but still it sometimes happens that the pipe will be plugged by being fused and great care is necessary.

It is best to have the tap-jacket made of bronze, but I see

no reason why a good quality of cast iron, or especially malleable iron, would not do, although it would be more liable to crack. There is a yoke cast on the front of the jacket immediately surrounding the tap-hole which is cored out for water space. The inside dimensions of this yoke are the same as the outside dimensions of the spout, and the yoke is to prevent the escape of slag and matte from between the spout and tap-jacket. A clamp is necessary so that when the two are put together they will

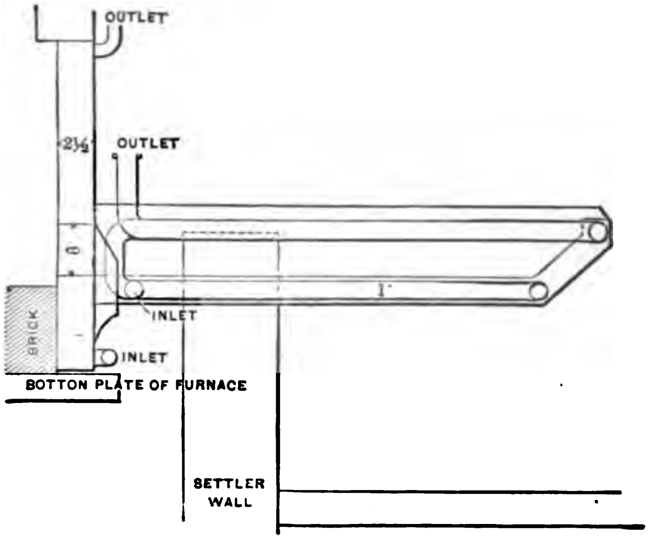


FIG. 7.—The Hixon Slag-spout; Longitudinal Section.

fit tightly and be easily separated to keep them from springing apart.

Spouts have been made in other ways, some of boiler iron, flanged and riveted together in such a way as to form one of much the same shape as that described, but the trouble is that there is too much cooling effect on the stream of slag flowing from the furnace, the result being that a skull forms in the spout which grows thicker rapidly, and finally the stream is interrupted entirely by the closing up of the channel. The furnace goes on smelting and the slag soon runs out of the tuyeres because it has no other outlet from the furnace. A spout with the coils, such as that described, exerts the least possible cooling effect on the slag-stream, and, if the furnace runs at any reasonable tonnage

and the slag is within the limits of composition mentioned on a previous page, such a spout will keep open and give no trouble at all from freezing up. A fire, either of wood or lump coal, should be kept on top of the slag-stream, and by this means the stream remains liquid and shows no shell or crust from the furnace to the end of the spout. Spouts constructed of boiler iron are a constant source of annoyance and will freeze up when the furnace is running at its best.

At Anaconda the settler was much smaller than at Aguas

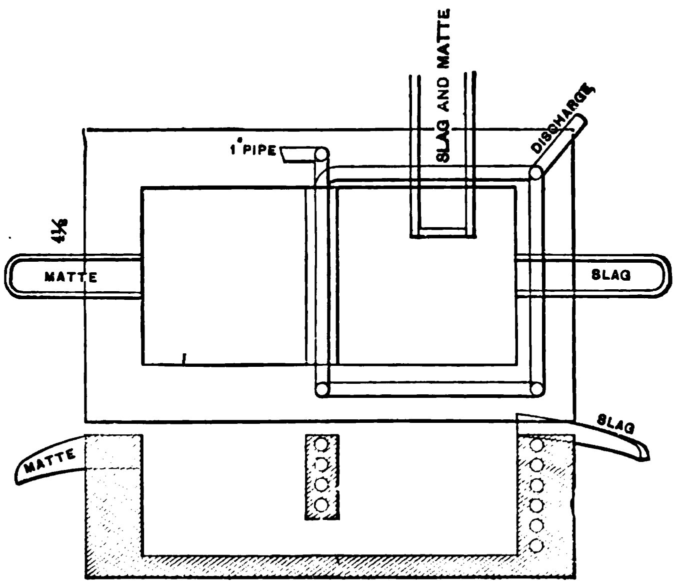


FIG. 8.—Plan and Section of Settler Used at Anaconda.

Calientes, and of a different type (see Fig. 8). The slag and matte were discharged together from the spout into the settler, where the matte by its greater gravity would go to the bottom and pass under the partition of pipe coil to discharge at a lower level than the slag at the opposite end. This partition and the lining of the slag-end of the settler were originally made of brick, but it was found that when the furnace ran a heavy tonnage the partition would be destroyed as well as the lining in the slag-end of the settler. Pipes with a water circulation were substituted and worked very well as long as the tonnage was kept up, but would chill the contents of the settler if the tonnage became

low, or the furnace was shut down for more than twenty minutes without tapping out. It is interesting to note that the copper contents of the slag discharged from these small settlers at Ana-

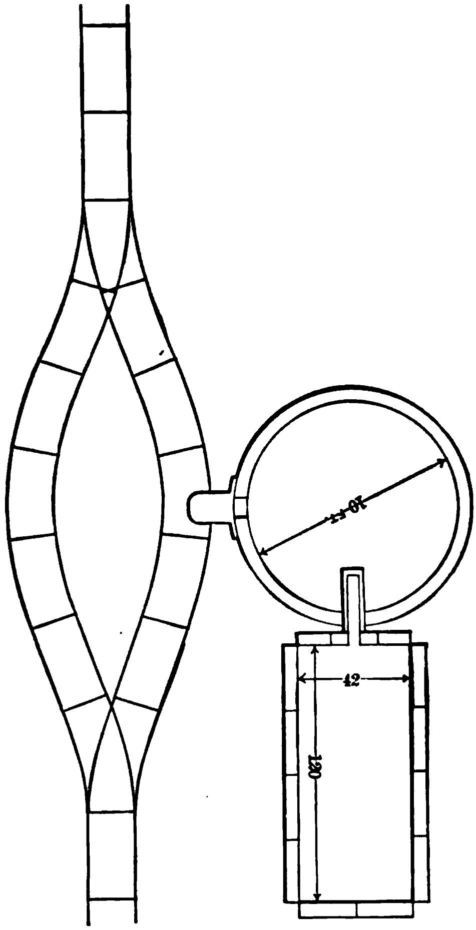


FIG. 9. — Plan of Furnace and Settler at Aguas Calientes.

conda were much higher than of slags made at Aguas Calientes, where the settlers (Fig. 9) were the largest in use (10 feet in diameter). On a charge producing 55 per cent. Cu matte at Anaconda the Cu contents of slag would average nine-tenths per cent., while under the same conditions at Aguas Calientes the Cu would not be more than five-tenths per cent., showing

that the capacity of the settler has much to do with the cleanness of the slag. With lower grades of matte the slag assays would be lower, but about the same relation existed in the two cases.

The jackets for a copper furnace of large size can be arranged in many ways, though there are but few good designs. In some cases the furnace is all one jacket from the crucible or base-plate to the feed-door, and the other extreme was embodied in the furnaces at Aguas Calientes, where there were twenty-four jackets on the furnace, counting the tap-jacket and spout. It is a very poor plan to multiply troubles unnecessarily, and especially water-jackets. The greater the number the more opportunities there are for one to leak or the water connection to become clogged and allow the jacket to burn out. On the other hand, it is not advisable to make the jackets too large. The middle course is the best. The front and back should be of one jacket each. The sides may be divided into any number to suit the conditions, but so that there shall be no seams or rivets exposed on the inside of the furnace. Jackets become very expensive as the size increases, on account of the unusual width of sheets necessary for the inside. Better results can be obtained by dividing the side of a 120-inch furnace into three or four jackets of equal dimensions. With three jackets to the side and two tuyeres to the jacket, there are six tuyeres to the side and twelve tuyeres to the furnace, which is quite sufficient, providing they are made 4 to 4½ inches in diameter. Four jackets to the side and two tuyeres to the jacket gives 16 tuyeres to the furnace, which is rather too many, but if they are made 3 to 3½ inches in diameter they will be quite satisfactory. The objection to a large number of tuyeres is that they will be so close together that the noses formed in front have a tendency to unite in a dark band all the way around the furnace, causing the zone of fusion to travel up. It is a mistake to have the tuyeres point downward, as is seen in some furnace drawings. The reason is that in case slag fills the tuyeres, as it is sure to do sometime in a long run, it cannot escape and solidifies there, whereas with a simple conical-shaped tuyere put in with its axis perpendicular to the face of the jacket, the slag can escape until the tuyere can be plugged and the trouble rectified.

It is also a mistake to put the jackets in a double row, one above the other, for the reason that whatever mud or scale there

is in the water will settle in the bottoms of the upper row, and that point being above the tuyeres where the heat is greatest will burn out, causing great trouble and unnecessary expense, to say nothing of filling the furnace with water and freezing it up. The writer has had experience with furnaces built in this way

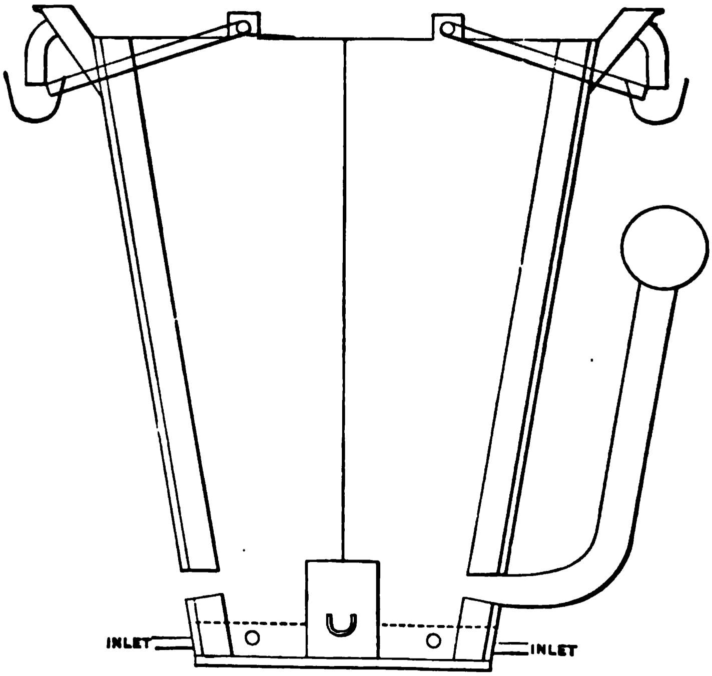


FIG. 10. — Design of Water-jackets.

and is thoroughly convinced that no greater mistake can be made than putting one set of jackets above another.

The hight necessary for the jackets to extend above the tuyeres depends upon how the furnace is to be run. If it is to be run on high blast and crowded to its greatest capacity, then the jackets should extend to the top of the charge, say 10 feet above the tuyere centers; but if there is to be low blast and easy running, as in lead smelting, then they do not need to be so high, since a shaft of fire-brick will serve quite as well. Generally speaking, the jackets for a copper furnace should extend from 8 to 10 feet above tuyere centers, which should be from 18 to 24 inches above

the bottom, making the jackets from 9½ to 12 feet in total hight, and of a width depending on the number used and the size of the furnace.

The jackets for the Anaconda furnaces were made of about the same hight as those for lead furnaces. This was found to be a mistake, as the burning out of the shaft proved. To correct it a series of coils of 2½-inch pipe were put in as a lining to the shaft of the furnace from the top of the jacket as far up as the brick burned out. The pipes were first put together with cast ells, but these broke and malleable ells had to be substituted.

There were three pipes to the foot and about six feet of the shaft was lined, so that it required about 500 feet of pipe and 54 malleable ells. The water was brought in at the bottom coil and made the circuit of the furnace eighteen times before escaping. This method of protecting the shaft is much cheaper than high jackets and, in case the water does not scale or contain much mud, will give equally good results. The method is mentioned here for the purpose of showing a way of changing a lead furnace to a copper furnace.

Under stress of circumstances, and in a locality whither it would be difficult to transport large jackets, a blast-furnace could be easily and cheaply constructed with a jacket of pipe 2½ to 4 inches in diameter in the same way as the brickwork was protected at Anaconda. Starting from the crucible or base-plate the coil should be put in place, each turn slightly larger than the one below, to give the shaft the desired taper of about one inch to the foot. When the level of the tuyeres is reached a nipple of 4 inches in length should be put in and the next coil raised by that amount, leaving a space 4 inches wide all around the furnace, which should be bricked up at all points except where the tuyeres are to enter. Above the tuyeres the coils should be extended to the desired hight and clamped securely together at the corners by means of stirrup clamps around the pipe, extending through an iron bar, and secured by nuts on the clamps.

The number of coils, malleable ells, and length of pipe to be used, depends on the size of the furnace, the diameter of the pipe, and the hight to which the jacket is to be carried. For a furnace 40 × 100 inches and a 10-foot jacket of 4-inch pipe, it would require approximately 600 feet of pipe and 100 ells. This would make a jacket that would do just as good work as the

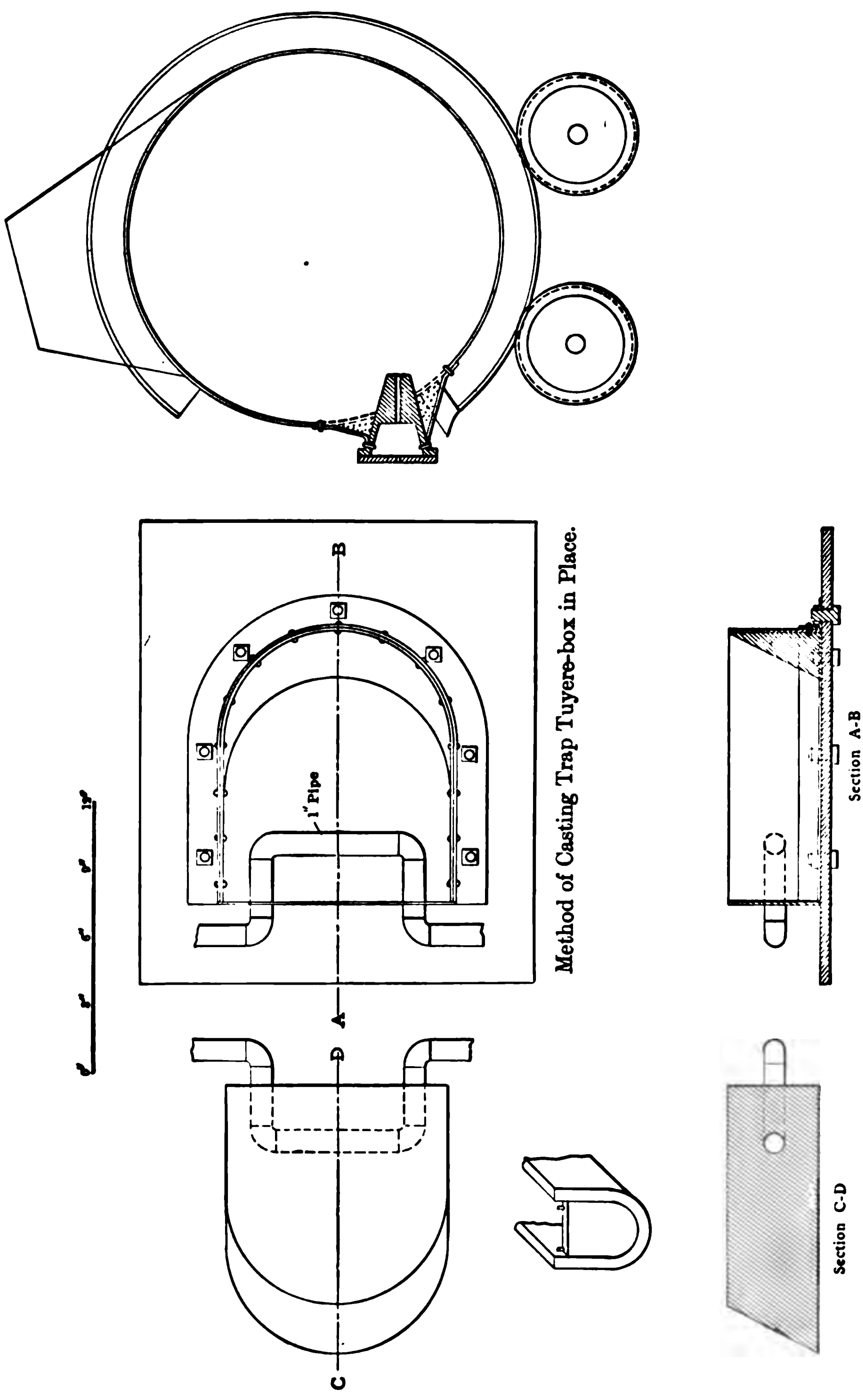
more expensive steel jackets of large size and great weight. The water should be brought in at the bottom and forced through the entire coil under a pressure of not less than 25 pounds. If this pressure is not obtainable, then the coil should be made in two sections, one above the other and supplied from a 6-inch main. The jacket made in this way will not be open to the same objection as if one steel jacket were placed above another, on account of the flushing out of any sediment by the rapidly flowing water.

Later Experience

The spouts now used on the largest copper furnaces are generally made of flange steel riveted together in a double shell, with water circulating between the shells, and a bronze tip at the end, to take up the wear of the stream of slag and matte falling from the spout into the settler. Some have been made by electric welding so that they appear seamless. The type of spout shown in Figs. 6 and 7 has been much in use on smaller furnaces, and with the sides of the spout raised to admit of another coil of pipe, it is as satisfactory as can be constructed in any other manner.

A change that improves the spout without changing the principle, is to construct it with the end open, like a trough, and then to have a separate block with pipe coil fitted into the cross-section, to act as a dam. This dam can also be constructed of quartz and clay, such as is used for converter lining; or it may be made of magnesite or chrome brick. In any case it will have to be renewed at intervals of from two hours to a week, depending upon how the furnace is running and what the dam is made of. The breast-jacket may be made in the same manner as the spout, by simply casting a pipe coil in a block of iron or copper.

At the works of the Tezuitlan Copper Co., the writer had made spouts for copper blast-furnaces, and tuyere-boxes for copper converters of converter copper; and these — I am informed — are still in use; it has been shown that they fully meet all requirements and are much cheaper than the same article as produced by manufacturers. The spout and trap for the blast-furnace can be made deep enough to suit any pressure of blast that may be used on the furnace. The spout as shown in the drawing is suitable for the pressure of 3 to 3.5 inches mercury or less. The hight of the trap in the spout can be varied



Method of Casting Trap Tuyere-box in Place.

Fig. 11.

by inclining the trap at a greater angle. We had no failures in casting these spouts; good castings were made each time. During the year the writer spent at these works, four spouts were

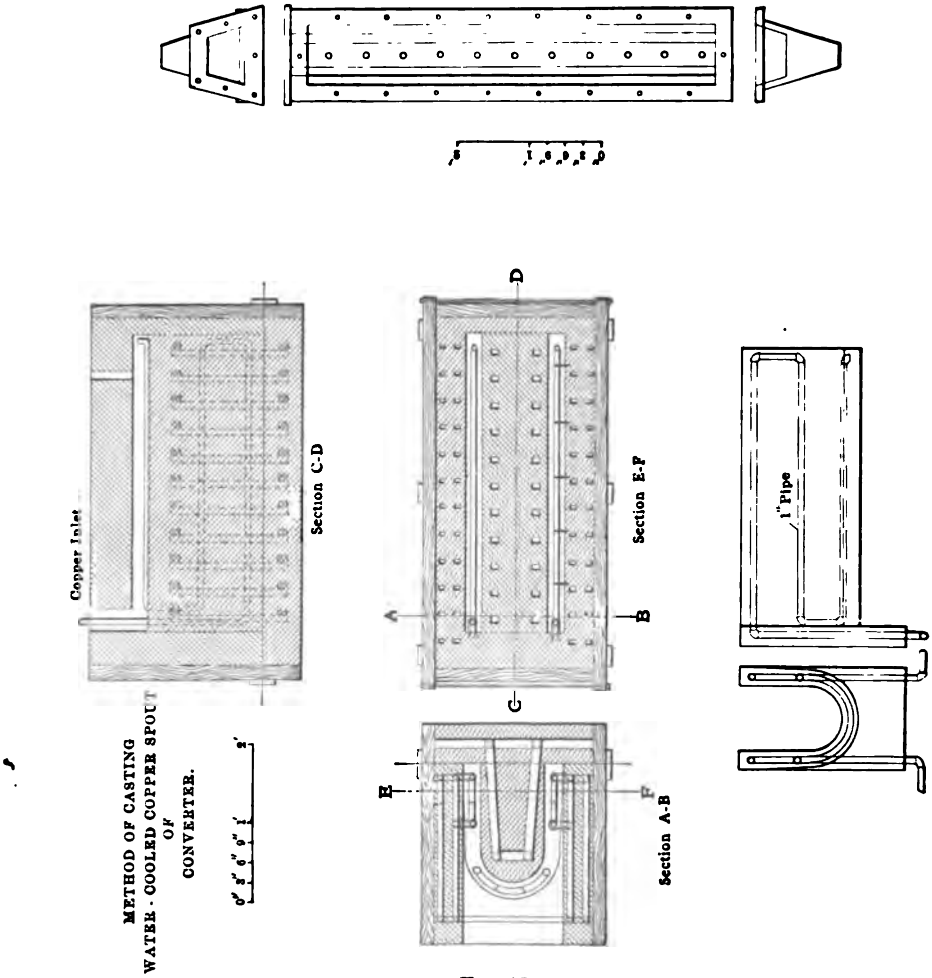


FIG. 12.

cast for use and three for reserve. The first one was cast too shallow for the blast-pressure, and after being used for some time another one was cast of deeper cross-section, and the first one was broken up and returned to the converters as scrap. This

spout with removable trap has the advantage that if the furnace is to be shut down, the trap can be removed while the blast is on, and all the molten material can be blown out into the settlers, avoiding the necessity for tapping the furnace on the side and having numerous explosions from contact with water.

A number of tap-jackets for the settlers were also cast in the same manner, curving a pipe around the tap-hole, casting a solid block of copper 15 × 24 inches and 3 to 4 inches thick; then boring out the tap-hole to the required size.

The copper tuyere-box was fastened to the shell of the converter in the manner shown. (See cut.) Two of these home-made tuyere-boxes replaced the tuyere-boxes originally supplied, which had been broken by contact with hot metal. The copper tuyere-boxes were in every way as satisfactory as the original, and were not subject to breakage. It was found that if a charge were run on the bare copper after the lining was destroyed, the copper tuyere-box would be melted as far back as the air-chamber. One such accident occurred, and it was repaired by turning the converter so that the tuyere-box was on top, putting clay on the inner side of the hole, taking off the outer cover of the tuyere-box and pouring in enough copper from another converter, by means of a ladle, to fill the burned-out portion, after which the tuyere-holes were again drilled.

Many articles for repairs about the plant, such as pump-glands and cams for shaking screens, were cast from converter copper and were found to give satisfaction.

Settlers

Settlers are simple steel tanks, with a refractory lining.

I have found that magnesite brick makes an excellent lining for a settler, and in order to economize brick it is best done as follows:

A bottom of converter lining about four inches thick is tamped into the settler; on top of this a course of magnesite brick on edge is laid, to cover the inside diameter of the settler. The wall of magnesite brick is then laid flat, one brick thick, in a circle on the inside diameter of the settler. As the wall is built up, converter lining is tamped between it and the steel tank. The converter lining prevents the radiation of too much heat,

and the 4½ inches of magnesite lining will last a year or more except in a small area nearest the slag discharge, and another area under the furnace spout, where the circulation is greatest. In these places the wall should be 9 or 13 inches thick. The magnesite brick must be laid in a mortar of magnesite powder, and the powder should be moistened with a saturated solution of magnesium sulphate. The reason for using the saturated solution of magnesium sulphate is that magnesite brick swells and decrepitates when wet with water. A settler lined as above was in use at the works of the Mond Nickel Co. for a year without change. Magnesite is a good conductor of heat, and if the whole lining is made of this substance it radiates too much heat for the furnacemen, beside inducing chills in the settler.

Care must be taken that the tank has sufficient strength to hold the contents in a fluid condition. The writer had an experience that will not soon be forgotten on account of a bursting settler. The Tezuitlan Copper Co. was using small cast-iron settlers of the Herreschoff type shown in Figs. 3 and 4, except that they were circular and water-jacketed all the way around. The water-jackets cracked and absorbed so much heat that the settlers would freeze up when the furnace was running its best. We desired to construct settlers ten feet in diameter of the usual tank type, and the heaviest sheet steel on hand was No. 16. Two thicknesses of this were riveted together and angle-irons put across the ends so that the settler was made up of sections of double No. 16 sheet steel bolted together, forming a ring standing on edge in a pan with an angle-iron edge holding the bottom of the rings. The settler was lined with converter lining and the furnace started. After running some days the lining got thin and the tank burst, allowing fifteen tons of matte and slag to flow equally in all directions. Some water on the floor added to the movement of things, but fortunately nobody was seriously injured.

Tap-Jackets

Nickel matte is much more corrosive on tap-jackets than ordinary copper matte, and after experimenting with all kinds of water-jackets, a block of magnesite was made by the brick makers of such a shape that it took the place of the tap-jacket. It was found to give the most satisfactory results.

Corrosive Action of Nickel Matte

As the percentage of nickel increases over copper in the Canadian ores, the corrosive action of the matte seems to increase to such an extent that the spouts and jackets ordinarily in use on copper furnaces become useless.

The ores of the Canadian Copper Co. contain more nickel and less copper than the ores of the Mond Co., and for this reason it has been necessary to abandon the use of the spouts and breast-jackets, copied from the plant of Tennessee Copper Co., and build a forehearth on the furnace, protected by cast-plate jackets and lined with chrome brick. With a forehearth and a settler on each end of the furnace, it is possible to tap slag from one end while the other end is being repaired.

Corrosive Effect of Water

The water in the streams of the Sudbury district is saturated with organic matter from peat bogs, and when it strikes the fire-sheets of the water-jackets it gives off carbonic acid. This carbonic acid in the presence of water acts upon the iron, corroding the sheets so that they leak in a very short time. On this account the Canadian Copper Co. has abandoned the use of ordinary steel jackets and constructs its blast-furnaces throughout of plate jackets with pipe coils surrounded by cast-iron in the manner shown in Fig. 24, except that they have the plates placed vertically instead of horizontally.

In 1907, in the transactions of the American Institute of Mining Engineers, there was a discussion regarding a mysterious corrosion of water-jackets reported from Douglas, Arizona. Considering the prevalence of similar conditions in places widely apart the writer has thought it worth while to republish here his contribution to the discussion:

From *Transactions American Institute*, September 28, 1907, *Victoria Mines, Ontario:*

I have had difficulties here similar to those encountered at Douglas [Arizona], and I found the cause to be the carbonic acid given off when the water was warmed. All the water in the streams in this country contains organic matter coming from peat-bogs and muskegs. It is brown in color, and when it strikes

the fire-sheets of the jackets, the carbonic acid is given off and travels up along these fire-sheets because of the bosh in the furnace. The lower side of the tuyeres becomes much pitted, and they leaked badly until I had copper tubes put in in place of iron ones.

The inner or fire-sheets were destroyed most rapidly opposite the cold-water inlet, where the greatest amount of carbonic acid was given off. Our boilers are not affected and are perfectly clear of scale. I think the acid is liberated in the feed-water heater, in which there are copper tubes, and after it is in a gaseous condition it does not attack iron, or at least the water is necessary to make it destructive. The pipes leading from the feed-water heater to the boilers are destroyed, but the boilers are not.

The Canadian Copper Co. had a purifying plant for the feed-water, and the pipe leading from the purifier to the different boilers went over them and each lead to the boiler came out of the bottom of the main pipe. Tests made of the water to the different boilers showed that the water to those nearest the purifier was much less acid than that to those at the end of the feed-pipe. In a casual conversation the superintendent spoke of it to me, and I suggested that the acidity of the water was due to carbonic acid dissolved in it, and that CO_2 gas had a tendency to enrich the water in the top of the feed-pipe, and, consequently, that drawn off for the first boiler from the bottom of the main, contained less acid than that which went to the last boiler.

I think the trouble at Douglas is due to a water-supply containing carbonic acid, coming from the deep wells, and this acid is probably due to the source of the water being something in the nature of a mineral spring, such as Saratoga, Manitou, or Apollinaris. Ordinary chemical tests would fail to detect any mineral acid, and the gas being small in quantity would escape detection.

The remedy for the trouble is to use copper fire-sheets, or to run the water through cooling towers and use it after the carbonic acid has escaped.

NOTE 1. Since the above was printed I have been informed by Dr. Douglas, that the trouble has been overcome by the use of a small amount of crude oil, in the jacket water.

From the *Journal of the Franklin Institute September*, 06:

CARBONIC ACID AS A CAUSE OF RUST

In view of the important place given to preservative coatings in the programs of engineering associations and the divergent views on the cause and prevention of rust, interest attaches to the data presented by Gerald Moody of the Central Technical College, at a recent meeting of the Chemical Society in London. The accepted theory has been that the presence of oxygen and moisture would always cause rust, and the view of some chemists that carbonic acid played an important part in the reaction has been considered disproved by experiments carried on by Dunstan, Jowett, and Goulding.

Mr. Moody, as reported by the London *Engineer*, held that minute traces of carbonic acid are sufficient to set up atmospheric corrosion, and he entered upon a series of experiments to justify his opinion. In these experiments extraordinary precautions were taken to exclude the minutest traces of carbonic acid. His plan was to keep a sample of highly polished iron in a few drops of distilled water for prolonged periods and to draw over it a continuous stream of air freed from carbonic acid by passage over caustic potash and soda lime. In some instances three weeks were expended in purging the apparatus of carbonic acid alone before the water was allowed to reach the iron, and for six weeks the pure air passed over the sample. At the end of this time the iron was as bright as when the experiment began. But, on the other hand, when air containing the normal quantity of carbonic acid was drawn over the sample, in six hours the bright surface was tarnished, and in seventy-two hours, during which time about sixteen liters of air passed over it, "the whole surface of the metal was corroded and a considerable quantity of red rust collected." The question presented by these experiments to the manufacturer of preservative coatings is how to exclude or neutralize the action of the carbonic acid carried by the atmosphere or by moisture.

Two correspondents of the *Engineer* comment on the above to the effect that there must be moisture to cause rust. One says that the experiment of the late Prof. Crace Calvert, in 1869 to 1871, showed that with dry carbonic acid there was no oxidation, and that the most rapid corrosion took place when the iron was exposed to damp "oxygen and carbonic acid." The other correspondent says that in all cases of rust water as well as carbonic acid and oxygen must be present; that no oxidation can take place without water, though it has not been ascertained how much moisture must be present or the precise part it plays.

VI

BLOWING-IN AND BARRING-DOWN A FURNACE

THE METHOD of blowing-in a lead furnace, commonly in use, is to fill the crucible with molten lead by first firing the furnace with wood for several hours to insure its being hot, and then throwing lead on top of the wood fire. A blast of air is blown down through the lead well or from a tuyere in front or back of the furnace to force the fire. By this means 250 bars of lead, or about the amount required to fill the average-size crucible, can be smelted in ten hours. This is generally done on the night shift, and the furnace cleaned of wood ashes by 7 A.M. A layer of fine wood or charcoal is then put on top of the lead to a hight of a foot above the tuyeres, a fire is started and the tap-jacket is put in place. The tuyeres are left open to admit air to the fuel, and after it has been thoroughly lighted and fire shows in all the tuyeres the furnace is filled up to the top of the jackets with coke. Slag charges are then put on alternating with about 12 per cent. fuel. After four or more slag charges alone are put on, then use one slag charge to one ore charge, or two slag to one ore, according to the conditions of charges, etc., until the furnace is filled up to about 10 feet above the tuyeres. A light blast is started and gradually increases to about one-half the regular amount, when slag is first tapped. From this time the blast is gradually increased for about four to six hours, when the furnace should be running on full blast, usually at a pressure of about 16 inches of water; but this pressure is dependent upon the fineness of the charge, and has to be varied according to the conditions governing the special case.

The barring-down of lead furnaces, to prevent the formation of blow-holes and consequent high lead loss, is made necessary by the increase of sulphide and zinc ores treated within recent years. In case the furnace is of the stack type, with side feed-doors, this can be done by the regular furnace crew in about one shift, more or less, depending on the condition of the furnace and

the energy and ability of the men. The furnace should be run down by the night shift so that the breast-jacket can be removed the first thing in the morning. The crust is then broken in, and the charge is raked out and removed. Two sets of men then start to cut out the shaft from above, while another set works below removing the crusts as they drop down the shaft.

After cutting down the wall accretions from above and removing all loose material from the furnace, a large hole is cut in the crust over the lead well, and the furnace is ready to resume operations again, with as good results as if just blown in. The wall accretions, if allowed to increase in thickness, will force the passage of the blast through a constantly decreasing space, resulting in greater losses both by flue-dust and volatilization. In case the furnace is of the top-feed type the work of barring-down is much greater on account of the greater distance the workmen are from the crusts to be removed, and in many cases to blow out such furnaces is found better than to bar down, for the reason that while the crusts can be removed in the shaft as far down as the top of the jackets, to work below that level with bars 20 to 22 feet long is exceedingly slow and difficult.

Barring-down in some cases is only partial, and is preceded by charging the furnace with coke to serve as a bed for the barrings to fall on. After barring in this way, which is generally practised in the top-feed furnaces, the furnace is filled up with charges, and operations are resumed. But it is not by any means so satisfactory as if carried on until the shaft is entirely clean down to the tuyeres, as is the case with side-feed furnaces, and is generally not done more than once in a campaign. In case the furnace is barred down completely, as first described, it is started in much the same way as if it were to be blown in afresh. A wood fire with coke on top, to the depth of two feet is made, then from five to ten bars of lead are added, according to requirements, to fill up the hole cut in the crucible, next a few slag charges and then the regular charge to the depth of about six feet, when light blast is turned on and gradually increased as the furnace is filled up.

To run a furnace down in the best way, either for the purpose of blowing-out or barring-down, it is necessary to reduce the blast, discontinue charges, and have on hand a considerable quantity of coke fines thoroughly wet, to be thrown in, a little

at a time, as the fire becomes too hot. In the case of a copper furnace which is running into a forehearth, it should be allowed to discharge into the forehearth until the stream from the spout becomes too small to continue to run, a bar should be quickly driven in the front and the furnace tapped on the side and allowed to discharge the remainder of the slag and matte until the blast is taken off. In cases of shut-down for any length of time the forehearth should be tapped and a bar driven in the matte-tap.

In the case of a lead furnace it results in too great loss of lead to run down to the tuyeres, although there may be four or five feet of wet coke fines on top; therefore the blast should be taken off about the time the charge is down to the top of the jackets and the remainder of the charge raked out on the floor.

It frequently happens from various causes that a crust will form over the lead in the crucible and force the lead out with the slag and matte. A sharp lookout has to be kept on the slag-pots at all times to prevent this, and, in cases where it is possible, to remedy it at once. The causes may be low lead charge, small matte production, bad reduction in the furnace, bad slag, water leaking into the furnace, or any one of a number of causes. The quickest way is to drive a bar through the tap-hole down into the crucible, but if the condition of the furnace is not changed by removing the cause, it will soon become impossible to keep the hole open by mechanical means. It is very seldom that conditions of charge and furnace are such that it is at no time necessary to drive a bar into the crucible.

It is stated frequently by men who pose as disciples of infallibility, that "you should not drive bars into the crucible." It is also worthy of comment that furnaces run by these same men have used up considerable steel and have developed some very good strikers. While not wishing to bear with unusual emphasis on this point, or to imply that it is necessary for a metallurgist to help his men by doing manual labor, still it is advisable in cases of this kind either to take hold with both hands or leave it entirely to the foremen and men.

The blowing-in of a copper furnace on a matte charge is not attended with as much difficulty as is the case with a lead furnace. It is important in the starting of any kind of furnace, that the water connections should all be looked after before the furnace is started, in order to provide for the increased use of water

during the short period of time before the jackets have become coated with a protecting layer of chilled slag. Before blowing-in a matte furnace the hearth as well as the settler, should be well heated with a wood fire for several hours, the spouts being thoroughly dried to prevent explosions by contact with matte.

After this it is only necessary to increase the wood fire in the furnace sufficiently to insure a thorough lighting of the coke covering, which should be put on to a depth of about 18 inches above the tuyeres. When the mass of coke is on fire throughout, charges of one-half matte and one-half impure slag should be put on with coke until the furnace is full. The blast is turned on and gradually increased until the full blast is reached. As soon as slag shows at the tuyeres, the bar is removed from the tap-hole, and the slag and matte are allowed to flow continuously into the forehearth. It is better to follow this plan of filling up the settler, especially if it be a large one, than to start at once on ore charges, because it insures the entire mass in the settler being in a fluid condition, and avoids the possibility of a hard matte-tap. If the matte and slag are not to be had for the purpose of blowing in, a charge should be figured that will produce the greatest quantity of matte together with a slag that shall run as easily as can be made from the ores to be smelted. In such a case the coke-bed should be somewhat thicker to insure the ores being smelted before arriving at the tuyeres. It is very important in cases where the settler is large that the slag and matte should enter the settler fast enough to float the crust that will form on top of the fluid mass. In order to assist this as much as possible, men should be stationed around the settler with bars long enough to be able to break the crust in a line next the walls of the settler, thus allowing the crust to float freely as the settler fills. The slag-tap should be closed to force the settler to fill up about six inches above its usual hight when running. This is done to raise the slag-crust into its proper position before allowing it to cool.

When it has risen to the proper hight the slag-tap should be opened very carefully and only enough slag allowed to escape, to keep the crust from rising still higher, until such time as it has chilled to a sufficient thickness to support itself when the fluid mass is tapped from under it. It may require four to five hours, and an occasional sprinkling from a hose, to allow the crust

to become thick enough. Some time before the settler is full, the slag and matte charges should be taken off gradually, and ore charges substituted, so that about the time the settler is full the ore charge should be down to the smelting zone.

Note from Later Experience

After the crust has chilled on top of the settler a coating of ground clay and quartz, such as is used for lining converter tops, should be put on to a depth of a foot. This will harden, forming a safe covering and preventing the radiation of too much heat.

VII

HANDLING BLAST-FURNACE SLAG

THE HANDLING of the slag from blast-furnaces is an important part in their management, and there are four methods now in use for lead furnaces and two for copper which have considerable merit. The first to be considered will be the methods in use by the Arkansas Valley Smelting Co. at Leadville, at the Omaha & Grant works in Denver, by the Mexican Metallurgical Co., San Luis Potosi, Mexico, and the Pueblo Smelting and Refining Co. at Pueblo, Colo. At each of these places one of the four methods for lead furnaces is in use and can serve as an example.

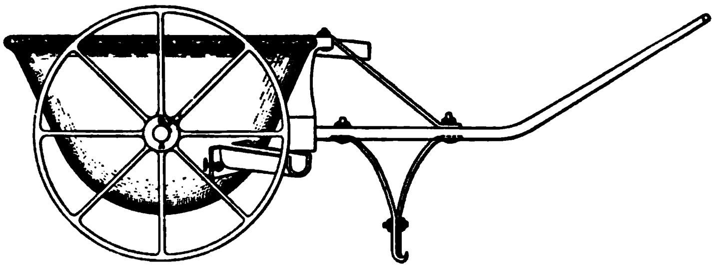


FIG. 13.—Matte-pots Used at Arkansas Valley Works, Leadville.

First, at the Arkansas Valley Smelting Company's works the slag and matte are tapped from the furnace into pots of the ordinary or Devereaux type and these are run on top of a reverberatory furnace built on a lower level, poured or partly tapped through the Devereaux hole into the furnace, where matte and slag are kept in a fluid condition until a complete separation takes place between them. The slag is then allowed to flow off into larger pots on cars, which are hauled away by an engine, while the matte, when it has accumulated sufficiently, is tapped from the furnace into beds. In this way the settling is an entirely distinct operation from the smelting, and one forehearth serves for several furnaces in blast.

The second method to be considered is in use at the Omaha & Grant smeltery in Denver (Figs. 14 to 16), where the slag is tapped from Devereaux pots into very large cast-steel pots of the same shape as the Devereaux, with the hole about two feet from the bottom, and another hole in the bottom for a matte-tap. These large pots are about five feet in diameter at the top, conical in shape, and about six feet deep. They are handled

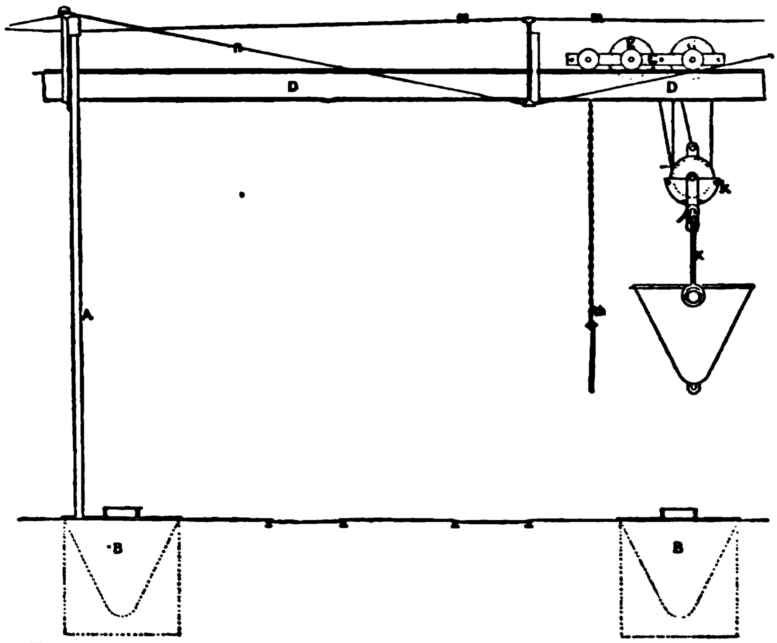


Fig. 15.—Matte Settling Pots Used at Omaha & Grant Works, Denver, Colo., Longitudinal Section.

by a crane and traveler, of very simple but effective design, and can be lifted about and changed with ease. When receiving slag from the Devereaux pots they are sunk in a pit in the dump so that the top of the pot is about an inch above the iron plates which surround it and serve as a floor for the slag-pots from the furnaces. Several small pots can be tapped into the large one at the same time, and two large pots being in use, one can be drawn off, while the other is filling. There is also a dumping-pot on a car, pulled by a horse, to take away the slag after settling the second time (Figs. 17 and 18). The plan of operations is to allow the

Fig. 14. — Matte Smelting Pots at Omaha Grant Works, Denver, Colo.

slag to settle as much as it will in the slag-pots, then to tap out the upper portion into the large settling-pots and allow it to settle again. When the large pot is full of slag it is tapped (as it stands in the pit) into the dump-pot on the car before mentioned, and this slag is hauled away to the face of the dump. If the large

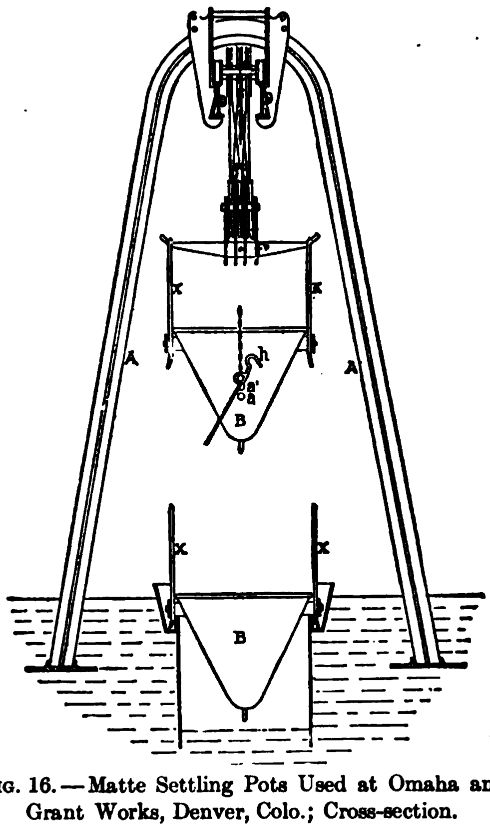


FIG. 16.—Matte Settling Pots Used at Omaha and Grant Works, Denver, Colo.; Cross-section.

settler has not enough matte in it to necessitate a change, it remains where it is and more slag is tapped into it after closing the Devereaux hole, and by repeatedly filling and emptying it of slag the matte that escapes from the small slag-pots accumulates until it is necessary to change the large settler. It is then hoisted and carried by the overhead traveler to a place provided, where it can be tapped from the bottom, and the matte allowed to escape into smaller pots or beds. Meanwhile another large settler is

put in its place, and the process goes on. The object of this second settling is to collect and recover that portion of the matte which at times will unavoidably escape from the tapping of the Devereaux pots.

The third plan, and the one practised at San Luis Potosi, Mexico, by the Mexican Metallurgical Co., necessitates the use of overflow pots at the furnace, of a size sufficient to collect the matte for from three to four hours, the overflow slag going into the Devereaux pots which are again tapped into other slag-pots of the same size, the contents of the latter being thrown over the dump. The overflow pots are emptied into cast-iron moulds

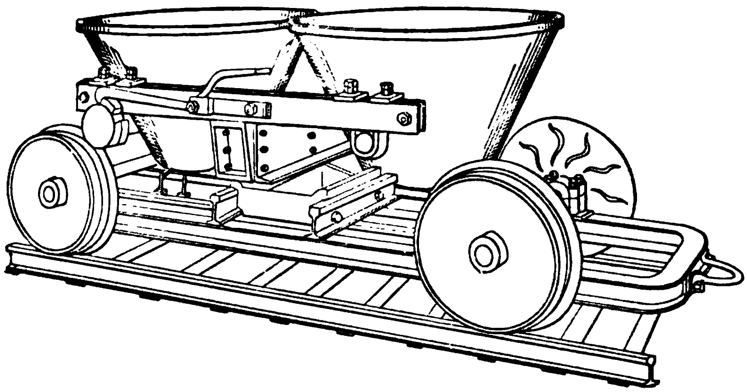


FIG. 17.—Slag-truck Used at Omaha & Grant Works, Denver, Colo.

arranged in a circle, so that a jib crane can be used to lift out the matte cakes when cool.

The fourth method is in use at the Pueblo Smelting and Refining Co.'s works at Pueblo. This makes the lead furnace a constant discharge, and the matte is tapped at a lower level than the slag, thus making a settler of the furnace.

At each of these places the visitor is given to understand that the method in use is better than all others, and it is a difficult matter to decide. However, each has points in its favor to recommend it, and it is to be supposed that the results are about the same.

In case a copper furnace is to be run with intermittent tap, the slag would be handled by one of the methods described for lead furnaces. With a constant flow and settler, the only two

methods in use are, first, by pots, which may be either of the ordinary kind on cars and run on tracks, or, second, by granulating with water. The first method needs no special mention except to state that the slag can be handled more cheaply by pots on cars than by the ordinary slag-trucks.

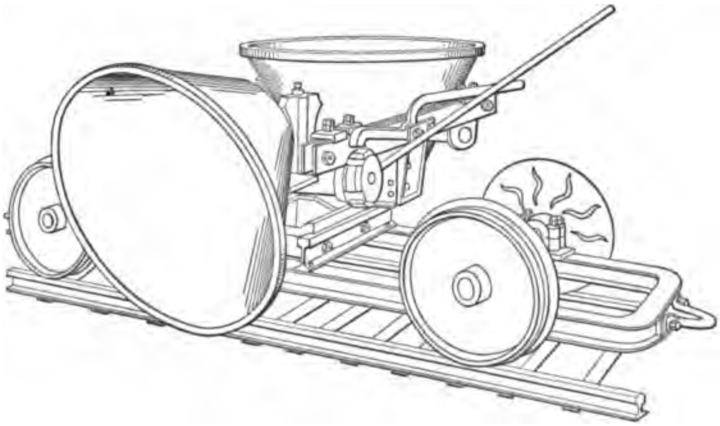


FIG. 18.—Slag-truck Used at Omaha & Grant Works, Denver, Colo.

The granulating and sluicing away by water necessitates a considerable amount of fall to the sluice, about 5 per cent. being the minimum, and about twice as much water is required as the jackets will use. A six-inch pipe with a pressure of 12 pounds will furnish enough water for the jackets and granulating slag for two 40 × 100-inch furnaces running 125 tons each day, the water discharged from the jackets being run into the slag-sluice.

Later Experience

In all large plants the handling of slag is now entirely by mechanical means, either by granulating with water, as in the case of the Anaconda smelter, or by large slag-pots on cars drawn by locomotive, as in the case of the Tennessee Copper Co. and the Granby Consolidated Co. For granulating slag a large supply of water is necessary under a pressure of 100 pounds or more. The plant must be located on a steep hill, above a large tract of waste land, a provision for the constant deposit of slag, day and night. Dump-room is filled up so rapidly that there is hardly a smelter in operation where this problem of the

future is not a source of worry. To effect the removal of granulated slag from a plant where the fall was insufficient, I have used the ejector described below.

Slag Granulating and Conveying Device[1]

The slag granulating and conveying device which has been in use for five years at the works of the Mond Nickel Company is illustrated in the drawing accompanying this article. The slag is granulated by a stream of water under a head of 20 feet issuing from a 3½-inch pipe with a nozzle ⅞ × 3⅛ inches. The water and slag are discharged into a cast-iron funnel connecting with the upper leg of a cast-iron Y. The lower leg of the Y is connected with a 5-inch water main from a cross-compound duplex pump, capable of supplying 1000 gallons a minute at 150 pounds pressure. The 5-inch main terminates in a manganese-steel nozzle of 1$\frac{5}{16}$ inches diameter. The jet from this nozzle strikes the granulated slag and water from the funnel, and drives it through the throat of the manganese-steel lining of the Y into the 5-inch manganese-steel discharge pipe. The discharge pipe carries the granulated slag and water to the dump, which is built out, and the pipe lengthened, as required. The discharge pipe may be level or stand at any grade upward or downward, depending upon the local conditions as well as the pressure of the water in the nozzle and the amount of slag to be discharged. In our case the pipe is on a slight grade, viz., 6 inches in 100 feet to allow for drainage in cold weather if the pump should be stopped.

The pressure on the nozzle will depend upon the amount of slag and water entering the funnel, and the length and diameter of the discharge pipe. If the discharge pipe is too large in diameter it will require more pressure on the nozzle to operate; for the reason, that, as the velocity of the water is reduced, the carrying capacity decreases as the sixth power of the velocity. Thus if the velocity is doubled the carrying capacity is increased as the sixth power of two, or 64 times as much. The experiments which led to the adoption of this method of disposing of slag were started with a centrifugal pump with a manganese-steel lining and impeller.

The small steam engine required to operate the pump could

[1] By Hiram W. Hixon, published in the *Engineering and Mining Journal.*

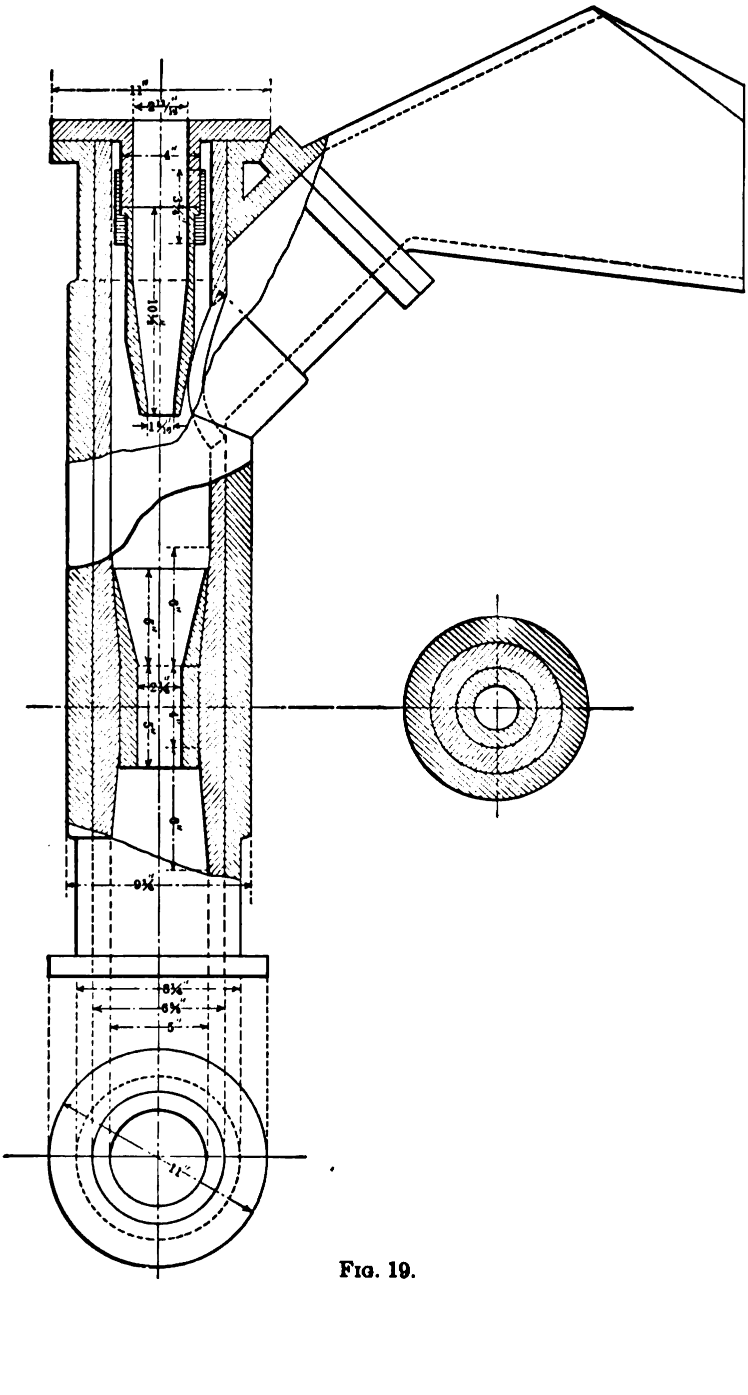


FIG. 19.

not stand the strain of high speed and constant operation, and soon broke down. During the time it was in use we found that when there was slag in the pump which filled up the space between the impeller and the lining, the discharge was satisfactory; but when the matte was tapped out of the settler and the slag stopped going into the granulator so that the pump was free of slag, it would not discharge the water alone and some water overflowed until the slag started again. The explanation of this is that the slag being so much heavier than water, the centrifugal action exerted a greater pressure on the discharge and made it act while filled with slag, but when only water was present the slip was so great that the pressure was not enough. Another condition that finally made scrap of the engine, was that when the matte was tapped out of the settler, the slag suddenly stopped going into the pump, which then automatically unloaded itself, and not having a governor it ran away, unless somebody happened to be near enough to shut off the steam.

The first experiments with the present conveyor were made with the use of ordinary 5-inch cast-pipe fittings. After proving that it would work, a special cast Y with the discharge leg longer than the other two was tried. This was made of hard iron, but wore out so much that the throat became too large and back currents were induced to such an extent as to destroy the nozzle and require much more pressure to operate. Manganese-steel nozzles and linings made by Hadfield, of England, were then tried and gave so much better results that now they are used exclusively.

The system does not require any special care, and two men on each shift are all that are required to attend a furnace smelting 250 tons in 24 hours. Some attention is necessary to the slag stream, where it strikes the water, but otherwise the system operates without any attention, so long as the throat of the lining is not worn out too large and the pressure of the water supplied by the pump, is kept up. Ordinarily 100 pounds pressure at the nozzle will carry the slag 400 feet from the furnace, and discharge a considerable amount of air, which is drawn in at the funnel with it. If the size of the dump increases beyond the limits of pressure that the pump can supply, a second Y can be put in at any point along the discharge and coupled up in series without the funnel, when the slag can be thrown out another 500 feet to the face of

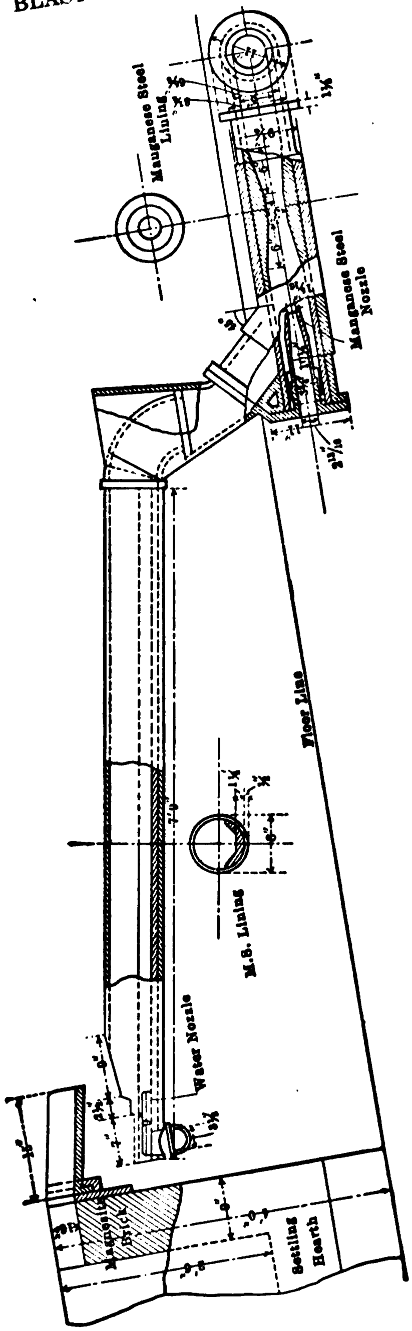

Manganese Steel Lining
Manganese Steel Nozzle
Floor Line
M.S. Lining
Water Nozzle
Magnesite Brick
Settling Hearth

FIG. 20.

the dump. At present the dump is being removed by a steam shovel for railway ballast, so that the second Y is not necessary.

For a small plant where there is plenty of water this is the cheapest method of removing slag, but in cases where water is scarce, or where several furnaces are each smelting a large tonnage, trains of large pots pulled by locomotive would prove more satisfactory. So far as I am aware this is the only conveyor of its kind in existence. The same principle is used to sluice out the ashes from the holds of steam ships, but they are intermittent in action and the water jet is supplied by the discharge from a steam injector. The Evans hydraulic elevator, used in placer mining, uses a jet acting through a vertical pipe to elevate gravel.

The wear on the discharge pipe of the slag-granulating system is mainly on the bottom; consequently it is turned over in order to cause the wear to proceed evenly over the interior before the pipe is discarded.

VIII

DESIGN OF LEAD BLAST-FURNACES

THE TENDENCY of recent years in constructing lead furnaces has been to increase the hight from tuyere level to feed floor as well as to increase the tuyere-section of the furnace.

The dimensions of a lead furnace at the tuyeres have no particular bearing on anything except the capacity; but the hight of the furnace above the tuyeres has a very decided bearing on the lead losses, as the experience with the type at Aguas Calientes will show. A simple statement of the facts as they occurred will be necessary to show how a lead furnace should *not* be constructed.

The plant at Aguas Calientes originally consisted of a concentrator of about 120 tons capacity, two roasters of a type that no one need desire to imitate, two copper blast-furnaces, 42 × 120, with forehearths 10 feet in diameter, and a copper converter 8 feet in diameter by 16 feet high. It was decided to add to this by building two lead furnaces. The copper furnaces were built up on a high pedestal of masonry so that a ladle could be run below the settlers in a position to catch the charge of matte when tapped for the converters. This made the foundation for the lead furnaces considerably lower than the copper furnaces. The charges for both sets of furnaces had to be elevated to the feed floor, and it was important that the charge floors should be on the same level, so that in case either elevator was under repair the other could be used for all the furnaces. The lead furnaces were accordingly run up until the charge floor was on a level with that of the copper furnaces. This made them 27 feet high from the furnace floor to the charge floor, or about 5 feet higher than top-feed furnaces are built in Colorado. There was nothing unusual about the furnaces except their hight and the feeding device, which was in imitation of the bell-and-hopper used with iron blast-furnaces.

On December 26, 1895, one of these furnaces was blown in on a charge containing 14 per cent. lead and a slag of about 33 per cent. SiO_2, 37 per cent. FeO, and 18 per cent. CaO. It started

off as furnaces generally do, with all tuyeres bright, slag hot, and producing lead a little too rapidly on account of the displacement of lead in the crucible by slag and matte accumulations.

The furnace was put on full blast about 6 P.M. and ran fairly well until about 4 A.M. the next morning, when the tuyeres had blackened and overfire started. The lead production ceased entirely by 12 o'clock M., and from that time until it was shut down, the furnace did not produce any more lead. The charge was changed many times; slag and matte charges were fed, and the blast was reduced in the hope of getting the overfire down. The bell-and-hopper feed was perfectly tight, and the furnace was closed up on top as tight as a tin box. The furnace continued to make slag and smelted a fair tonnage of ore, but produced no lead. The writer was of the opinion that the bottom of the crucible had sprung a leak and the lead was going into the foundation.

The furnace was then shut down and the other furnace, which had been put in readiness, was blown in. The charge used did not differ greatly from that put on the first furnace, though the fuel was increased from 12 to 14 per cent. and the blast-pressure reduced; but in spite of these precautions, and the most careful attention, the furnace got hot on top and the lead production stopped entirely in about 24 hours after the furnace was started.

From this point until the furnace was blown out four days later, all the changes that could be suggested were tried. The slag was made acid and then it was made basic. It was run high in lime and low in lime, but no lead came out of the furnace. Bullion was fed back again, and it seemed to disappear as completely as if it had never existed.

The lead gradually fell in the well and bullion was melted in the pot and poured into the well until it was full, when it would gradually disappear again, showing either that there was no lead going into the crucible or that the crucible was leaking. After a council of war it was decided to blow the furnace out, as too much money was being lost to allow this state of affairs to continue.

Accordingly it was decided that another man should try to run them, and Dr. Charles Harbordt was sent there to do the metallurgical work. The doctor is a valued friend of the writer, and was offered every assistance that the light of past experience

could give as to the running of these furnaces. One of them was blown in, and to describe its working would only be a repetition of what has been said of the other two attempts, except that it ran a week instead of four days without producing any lead. The furnace was then blown out and a message sent to headquarters that no more furnaces would be blown in until the general superintendent, who was the designer of the plant, came down. When he arrived on the scene he blew in one of the furnaces. After a short trial he removed the bell and hopper feed and had the furnace fed by hand through a small hole; under this condition lead was produced for three days, but in a rapidly decreasing proportion to the amount on the charge.

The second furnace was blown in and acted the same as in all previous campaigns, producing a little lead for two days and then the lead production became nil. The tuyeres would become black and hard, and frequently raw ore could be found at the crucible. Shutting off the blast from the tuyeres would have the effect of burning off the nose after several hours, but it would only take them as many minutes to become black again when the blast was turned on once more. Besides, the furnaces were not being run to keep the tuyeres bright but to produce lead, and as to this point there could be no question — they were a failure.

The bell-and-hopper feed was withdrawn from the first furnace, and it was again blown in, all the ores and coke being thoroughly soaked in water. Frequently the hose was used with good effect on the feed floor and gradually the zone of fusion was brought down to the tuyeres. The lead production, which up to this point had been nil, gradually rose until about 60 per cent. of the lead on the charge was produced as bullion. This was the best that could be done. When the bell-and-hopper feed was again replaced the lead production ceased as quickly as if there had been no lead contents in the charge.

To a metallurgist who had not seen these things as recorded this might seem to be an exaggerated statement. It would appear to be impossible to lose all the lead in the charge if so much as 14 per cent. were used. And that is exactly the way it looked to the writer when the first two furnaces were blown in, but after seeing three more in blast and under the supervision of men who have had long experience in lead smelting, he became convinced that science was indebted to the designer for finding out two things:

First, that a lead furnace closed in on top will soon show overfire, and the lead will be wholly or partially lost as volatilized fume; second, that increasing the distance between tuyere level and feed floor beyond the proper hight of the smelting column, has the same effect as closing the top of the furnace. It assists in volatilizing lead by preventing the cold air which is drawn in above the charge from cooling the top layers of charge.

After making many attempts to correct the losses it finally became necessary to tear out the crucibles, cut off the shaft of the furnace to about the standard hight, 22 to 23 feet from the furnace floor to charge floor, throw out the automatic feeding device, and return to feeding with a shovel.

It is certainly of great value to the profession to know what cannot as well as what can be done, and while this information is all of a negative character, still in many respects such an example has its uses. While it was certainly very annoying and perplexing at the time, it is all clear now, and the writer is very glad to be able to impart the experience to others even at the risk of being suspected as a party to the error.

After blowing out, a sample of the bullion remaining in the crucibles was taken, and it was found to assay about 400 ounces Ag per ton, while according to the charge calculation it should only have had 180 ounces Ag, thus showing that the lead had been volatilized in much greater proportion than the silver, and that there had been a cupelling action in the furnace.

After a careful sampling and weighing of all products and giving credit for the lead in the slag produced, which was all high in silver and lead, it was found that 63 per cent. of the lead and 23 per cent. of the silver were unaccounted for.

The deductions that can be drawn from this failure are very simple and exceedingly important. They point clearly and with force to the fact that the first lead furnaces in use in this country were better adapted to the saving of lead than the modern high-shaft, top-feed furnace. The furnaces with the charge doors on the side, a stack above the floor, with downtake from this stack to the flue beneath the floor, as originally constructed in Leadville and at the Colorado Smelting Co.'s works in Pueblo and also at Great Falls, Mont., nearly fulfil the opposite of all conditions that existed at the furnaces at Aguas Calientes. In the first place, the amount of flue dust and the loss resulting therefrom

is much less in a stack furnace than in a top feed, for the very sufficient reason that the flue dust must rise a distance of 10 to 14 feet before entering the downtake of a stack furnace, and, on the other hand, in a top feed the downtake is right at the top of the charge, where every inducement is offered for fines to go into the flue and thus heavily increase the mechanical loss.

In a stack furnace the air entering the charge doors comes into contact with the top of charge, thereby preventing considerable lead losses. That the cooling of the top of the charge in the furnace by the air drawn in at the charge doors has this effect, is proved by the excessive losses in the furnaces at Aguas Calientes, where the only points of difference were bad feeding and excessive hight of shaft above the top of the charge, which prevented the cold air having access to the charge. The downtake was immediately below the charge floor, and the furnaces were fed at various depths from the downtake to thirteen feet below the charge floor. They produced no lead when the smelting column was 18 feet above the tuyeres, and did the best work when it was 10 to 11.

This experience, disastrous though it was, was one of the greatest object lessons ever given to lead smelters. It shows plainly that lead has to be treated as a very volatile metal, closely resembling mercury in its behavior; and that if the greatest possible saving is to be made, the blast-pressure must be light, the smelting column not much over 12 feet above the tuyeres, and the greatest possible amount of cold air must be allowed to enter the furnace at the top of the charge. If these points are well taken, then all the top-feed thimble furnaces that have been built so extensively in the last ten years, are steps in the wrong direction.

There is another point about the stack furnace that is much in its favor, and that is, being fully 5 feet shorter from the charge floor to the furnace floor, it is much easier to bar down when accretions form and, by doing this once a month, can be kept in blast for any length of time desired.

No doubt there will be many to disagree with these conclusions, and the writer will admit that he was of different mind until the force of experience brought out the points in question.

Briefly stated, the points of superiority of the stack furnace over the top feed are:

(1) It makes less flue dust; (2) it runs cooler on top; (3) it loses less by volatilization; (4) it is more easily barred down; (5) it can be kept in blast longer.

Many kinds of thimbles and charging devices have been introduced and used on lead as well as on copper furnaces, and they have as often been discarded, the method of feeding with the shovel into an open-top furnace thus far being found superior to mechanical feeding for many reasons.

A furnace does not run alike in all parts and requires to be fed the greatest amount at the point where it sinks the most rapidly. Certain kinds of material must be put in particular places in order to correct irregularities of running. Any approach towards closing in the top always has the effect of drawing the fire up, and results in increased losses by volatilization; and this loss by volatilization, in case of lead, will increase rapidly as the air is prevented from entering at the top of charge.

The best method of feeding is to have a fender in front of the charge door (to prevent as much as possible the dumping of charges into the furnace either accidentally or intentionally, but more often the latter) and to scatter the charge thoroughly over the surface of the coke which has been shoveled in in the same way, but with more on the sides near the furnace walls than in the center. The reason for putting the coke next the furnace walls is that as it goes down it may burn off accretions, and when it has arrived at the tuyeres the heat will be driven into the ore. Sometimes accretions of considerable size can be taken from the walls of the furnace by persistently feeding the fuel against them. From the very nature of things it will be apparent that a chance distribution of the charge, as would be the case with bell-and-hopper, is the poorest possible contrivance for feeding a lead furnace. It worked on the copper furnaces at Aguas Calientes, though very unsatisfactorily, — principally because it was impossible to keep the furnace cool on top, the result being high silver losses and the consumption of the fuel before it arrived at the proper smelting zone.

IX

LEAD SLAGS AND LOSSES IN LEAD SMELTING

THE CALCULATION of slags for lead furnaces is carried on in exactly the same way as for copper work, with the additional care that has to be taken to keep a sharp lookout for the zinc, sulphur, and baryta. The ores have to be bedded in such a way as to admit of their being used to the best advantage, and this is an absolutely necessary precaution to insure success. Frequently it may be of advantage to change the charge, to put on more or less of some class of ore, either lead, dry or sulphide, and unless the ores are bedded according to these classifications it might be difficult to make the required alteration.

The following is a fair example of lead charge as made up from beds:

	Weight	Per cent. SiO_2	Pounds SiO_2	Per cent. FeO	Pounds FeO	Per cent. CaO	Pounds CaO	Per cent. Pb	Pounds Pb	Per cent. S	Pounds S
Carbonate	400	26	104	31	124	6	24	12	48	4	16
Silicious lead	200	41	82	12	24	2	4	10	20	6	12
Roasted.........	100	15	15	35	35	0	0	31	31	5	5
Iron ore.........	100	10	10	55	55	2	2	2	2	..	..
Lime	200	3	6	..	..	52	104	..	..	..	..
	1000	..	217	..	238	..	134	..	101	..	33

Owing to the presence of some lead and zinc in lead slags, the sum of the SiO_2, FeO, and CaO do not, as a rule, amount to more than 88 per cent., so that figure has been used in this calculation. Also, it has been assumed that 20 pounds of the 33 pounds of sulphur on the charge will go into the matte while the other thirteen will be burned off as sulphurous acid. Twenty

pounds of sulphur would indicate about 80 pounds of matte, as lead mattes carry about 25 per cent. S, and the iron equivalent of 40 FeO, so that 40 per cent. of 80 pounds equals 32 FeO, and that amount should go into the matte, leaving 206 pounds of FeO for the slag.

This type of slag, 34 SiO_2, 33 FeO, 23 CaO, is a favorite with many smelters, and is largely used in the smelting of copper as well as of lead, although in copper work much less lime will do just as well and result in treating more ore. With only 33 pounds of sulphur on the charge the matte formation would in all probability not be as much as 80 pounds, and very likely not more than 50 or 60; if the lower figure should prove to be correct, then only 20 pounds of FeO should be deducted, which would change the resulting slag composition to 33.4 per cent. SiO_2, 34.1 per cent. FeO, and 21 per cent. CaO, which is just as good a slag as the first, and whichever is used, there would be no material difference in cleanness of work. Slags as low as 30 per cent. SiO_2, and as high as 40 per cent. FeO and 20 per cent. CaO, are perfectly safe, and formerly, under certain peculiar conditions which now unfortunately do not exist, were good commercially.

Generally speaking, iron slags are more fusible than lime, and as a consequence give rise to less loss in the volatilization of lead and silver, and they will absorb more zinc without becoming dangerously hard in the tap-hole.

The peculiar feature of high-lime slags is their remarkable flint-like toughness when they once get cold. A furnace running on such a slag may be perfectly free and in good condition, and fifteen minutes later, owing to a hard breast may have slag running out of half the tuyeres.

The writer has in mind one superintendent who if he has one particular hobby more than another it is to run on high lime. No matter what is to be accomplished, it can only be done by raising the lime. If the reduction is to be improved, the fuel cut down, the furnaces run faster, or what not, just raise the lime and, presto, you have it. Another peculiarity in certain minds is that by some optical method of analysis the lime is always higher than a chemist can get by any known method except the "graphite."

It is certainly a bad plan for a metallurgist to insist on having the slag analyses agree with his calculations, and to insinuate

that the chemist does not know what he is about simply because he reports the results as he gets them. It might appear that this is totally superfluous, but the writer has lived in the same atmosphere with a great many men who would forget all the possibilities of mistakes that could be made in weighing charges, in bedding ores, of throwing on the wrong ore at the scales and a shovelful more or less, and then blame the chemist for inaccuracy of work. Besides, slag as it comes from the furnace is not of a constant composition. Every pot is slightly different from every other pot, and a close approximation is all that can be expected.

The losses in lead smelting are generally estimated at 5 per cent. Ag and 10 per cent. Pb, and in purchasing ore it is customary to deduct this amount from the metallic contents of the ore in making payment. The actual losses, or such as are shown in statements, vary according to local conditions of charge, plant, and ability of the metallurgist.

With three unknown quantities in the equation, each of which is subject to great variation, it is not surprising that the known results should vary considerably.

Properly speaking, the losses in lead smelting can be kept within the allowed limits of 5 per cent. Ag and 10 per cent. Pb, but to do this it is necessary that the furnaces shall not be run on too low a lead charge, say not below 11 per cent., or the tonnage crowded too much by high blast.

The losses of lead in the slag will increase as the amount on the charge decreases, for the reason that there will be more slag, and with the same contents of lead it would represent a greater percentage of loss. In addition as the lead on the charge decreases the charge becomes more infusible, and as a consequence a greater amount of lead is lost by volatilization.

The writer has had returns from months during which the results on accurate charging of all contents represented a loss of 4 per cent. Ag and 8 per cent. Pb, and in other months 7 per cent. Ag and 14 per cent. Pb. At all of the Colorado smelteries there is an effort to crowd the tonnage of the furnace as much as possible, and this can only result in heavy losses of silver and lead. There is also considerable roasting and slagging of sulphides as a preliminary step to smelting, and it is safe to say that 5 per cent. of the silver and 15 per cent. of the lead contents are often lost

in this one operation alone. If the ores are simply roasted the loss will be very much lighter, probably not over 2 per cent. Ag and 4 per cent. Pb, but the slagging or fusing of the roasted charges in the fuse-box, as practiced at many of these works, is certainly productive of very high losses in the general return. Accurate returns from slagging a lot of 50 tons of flue dust at the Arkansas Valley smelting works, in Leadville, showed 11 per cent. silver and 25 per cent. lead loss. Similar tests on slagging ores containing no lead, showed 5 to 8 per cent. silver loss in roasting and slagging, while the simple roasting loss was below 1 per cent. If any considerable portion of the tonnage of a smelter is treated in this way, the smelting losses cannot be expected to be within 5 and 10 per cent. limits. There is another feature that plays an important part in smelting losses and that is the amount of by-products or matte, barrings, and flue dust produced. If the contents of these by-products are credited to the smelting account at any more than 95 per cent. of the full amount, an error is made, for they have to be smelted again, and it is certain that in doing so 5 per cent. of the silver and 10 per cent. of the lead contents will be lost.

The percentage of fuel necessary to do good work or to smelt different classes of ore at different plants is subject to great variation. The simple fact that a furnace can be run on 9 to 10 per cent. fuel basis is not proof sufficient that the addition of 2 or 4 per cent. would not smelt enough additional ore to make it a better commercial proposition. For example, at Denver, Pueblo, or Leadville, with coke at $6, and the average charge margin at $4, a furnace that would smelt 40 tons a day on a 10 per cent. fuel charge, would probably smelt 45 tons on 12 per cent. fuel. In the first case it would use 4 tons coke, $24; margin, $160. In the second it would use 5.10 tons coke, $30.60; margin, $180; showing a gain of $13.40 per furnace in favor of higher fuel. This is only true within certain limits, however, because the increase of fuel beyond the limits required for smelting and reduction does not increase the tonnage, but rather decreases it, owing to the extra time required for the combustion of the surplus before the charge can come down.

The effect of different kinds of coke on the running of the furnace as well as upon the losses is marked, although the reason is at times obscure. There is a coke made at Sabinas, Mexico,

which, according to reports of metallurgists who have used it at Monterey and Velardeña, causes an abnormal loss of lead in the blast-furnace, owing probably to high percentage of ash and volatile matter; the volatile matter passing off as gas causes over-fire, which in turn volatilizes lead. A coke may have as much as 20 per cent. ash and if well baked and free from volatile matter may yet produce fair results in the blast-furnace, although a coke with less ash will do proportionately better work.

It is also customary to increase the fuel according as the altitude of the place is higher. At Leadville the fuel consumption is from 3 to 5 per cent. more than at Pueblo, 5000 feet lower. The reason for this is that a greater quantity of air has to be blown through the furnace to get the required amount of oxygen, and the surplus of inert nitrogen produces a chilling effect on the furnace, which has to be remedied by the addition of more fuel.

X

NOTES OF IMPROVEMENTS AND CHANGES IN LEAD SMELTING IN 19[illegible]

Mechanical Roasting-Furnaces. — Since the first edition of this work experiments with various mechanical roasting-furnaces have been continued. Attempts have been made at some customs works to roast sulphides high in lead in furnaces adapted only for roasting iron and copper sulphides, resulting in only a partial elimination of the sulphur and a product unfit for the blast-furnace. Brückner furnaces are now so operated that any class of material containing lead or zinc can be roasted and sintered to a proper condition for blast-furnace smelting.

The success of the operation lies in having the furnace lining sufficiently thick to prevent radiation, and in revolving the furnace slowly to allow time for sintering, as it is necessary to reach the sintering temperature before the lead and zinc sulphates can be decomposed and the sulphur eliminated. If copper-iron sulphides are treated, a more rapid rotation, about once in 40 minutes, is correct practice; but if zinc-lead sulphides are roasted there should be but one revolution in 1¼ hours. Proper manipulation consists in firing to a roasting heat when the furnace is charged, keeping up this temperature for a period of 24 to 36 hours, in the latter portion of which the temperature is gradually raised to the maxium obtainable in the fire-box attached. Sintering takes place in the latter 12 hours, the entire period being 48 hours. This will produce roasted material that should contain between 3 per cent. and 4 per cent. sulphur on a similar character of charge with the same lead contents that is treated in calciners. The cost for roasting in Brückners is about one-half that in the calciners, while the losses are supposed to be about equal. The amount of flue dust produced by Brückners depends entirely on the period of rotation and the fineness of the charge. The reason for the slower rotation in roasting mixed sulphides is that when

the Brückner is rotated more rapidly, the charge is turned under before it is heated enough to eliminate the sulphur.

Owing to the plastic condition of the partially sintered charge a portion of it adheres to the lining, and after roasting four or five charges the furnace should be cooled down by spraying, and the accretions dug out. The length of the Brückner has been increased to 28 feet, it having been found that the fuel consumption is less per ton of ore treated, as with a shorter furnace heat which might have been employed in roasting ore is wasted in the flue.

The limit of sulphur contents of a roasting ore depends on the ore supply of the works that are to treat it, the roasting capacity of the works, and the cost of roasting. With a high roasting cost, a limited roasting capacity and ores low in silver and gold, it is, at times, advisable to raise the roasting limit to as high as 10 per cent., for the reason that a 10 per cent. ore, provided 70 per cent. of the sulphur goes into matte, will produce only about one-third of a ton of matte. It is cheaper to smelt direct, produce this one-third of a ton of matte, and roast it, than to roast the entire tonnage of the ore. At times, if the ore supply is such that all the ores contain sulphur, it may become exceedingly hard to run the blast-furnace and produce a reasonable amount of matte. Under these conditions it is better to roast ores of less than 10 per cent.

If the first cost of roasting is low, and the roasting capacity is sufficient, then it is better practice to lower the roasting limit to about 6 per cent.; but local conditions govern these things to such an extent that no hard and fast rules can be laid down. If the ore supply of a smelter is entirely oxidized ores, smelting becomes very difficult, and the metallurgist is glad to receive a limited amount of sulphide ores which produce matte, as it has a beneficial effect on the running of the furnace. Under these conditions, ores as high as 25 per cent., or 30 per cent. sulphur are smelted without roasting. It appears that while the production of matte is desirable up to a certain point, beyond that point it becomes a nuisance; while if no matte at all is produced, the metallurgical work suffers from that also, and the condition of the furnaces is by no means as good.

The effect of coals with varying percentages of ash on roasting is remarkable. There is a wide variance at times, owing to

bad separation of the bone, when the sulphur determinations immediately show an imperfect roast. With coal of high ash it is impossible to produce good roasted material in Brückner furnaces or in reverberatory calciners.

Utilizing the Brückner's Waste Heat. — I have made experiments in utilizing the heat of the waste gases of the Brückner flue in making steam, and have succeeded in raising steam to 100 pounds pressure in small quantity. The steam generator is a series of ten 3-inch pipes 30 feet long, suspended from the arch of the flue by hangers, and all connected at one end with a common-feed water-main, and at the other to a steam drum. Ample return pipe is provided for the water which is blown into the steam drum to flow along with the feed water. The pipes are soon coated with flue dust and do not absorb the heat as rapidly as they do when clean. The steam generated falls off rapidly and, unless the pipes are cleaned, will amount to little. This device, acting as a feed-water heater attached to a boiler, effects a considerable saving.

Peculiar Effect of Coke in Smelting. — I have found that two cokes, differing merely in physical appearance, will give widely differing results. Coke manufacturers will, at times, send out coke that does first-class work, producing slags and mattes low in lead. Another shipment will be received from the same manufacturers that, without any apparent difference, either in ash or physical properties, will do poor work. I have experimented with two furnaces running on the same charge with different cokes, the results of which showed great differences of lead contents in the mattes and slags produced by the two cokes. The coke has then been changed on the two furnaces, and the conditions have been reversed with the change of coke, showing that it was unmistakably the effect of the latter. I have investigated porosity, density, specific gravity and coke analyses, all without being able to determine the cause.

Later Experience

The difficulty in this case proved to be in the size of the pieces of coke and the effect of feeding too much of the charge in the center of the furnace. The coarse coke and large pieces of ore, slag and limestone rolled to the walls and made lines of

least resistance which, as before mentioned, caused overfire and bad reduction. When the coke was crushed to potato size and the charges dropped against the walls, forcing the coarser pieces to fall to the center of the shaft, this mysterious behavior of the coke ceased. The comparative value of cokes can be determined by the number of grms. of lead that a grm. of coke will reduce from a charge of litharge fused in a crucible.

Liquation of Bullion Drosses. — The smelting of lead ores containing some copper, produces a great deal of dross on the bullion, which must be skimmed off before pouring into the moulds. This dross was formerly thrown back into the furnaces, but the resmelting of a large quantity of lead in the furnace is by this means unavoidable, and the loss by volatilization is apt to be larger than the expense of liquating; therefore, it is now liquated in a small reverberatory furnace with an inclined hearth. The remaining copper dross, which is found to run from 20 per cent. to 30 per cent. copper and from 30 per cent. to 50 per cent. lead, is then removed from the hearth and resmelted with copper matte.

Effect of Slag Composition in Smelting Zincose Ores. — With a proper composition of charge, zinc ores may be smelted without ill effects in the blast-furnace. This is done by increasing the iron and lowering both silica and lime. If the zinc is high the slag should not run over 30 per cent. SiO_2, with 35 per cent. FeO, and about 16 per cent. CaO. If an attempt is made to carry high silica with high lime or zinc, the tuyeres grow hard and the blast will not enter the furnace, while a mushy, spongy mass, neither matte nor slag, separates from the matte, collects about the tap-hole, and refusing to flow out, brings about the slagging of the tuyeres. At the same time the settlers close up and the capacity of the furnace is so reduced that smelting is no longer a commercial success. I have, at times, had as much as 15 per cent. zinc oxide in the slag, the furnaces running in good condition with a fair tonnage, producing slag and mattes low in silver and lead; but if the silica, for any reason, became as high as 34 per cent., the tonnage would rapidly fall off, the tuyeres get hard, the settlers close up, and the slag assays become high.

Mechanical Charging Apparatus.— The writer has constructed and has in successful operation an apparatus for charging furnaces (Figs. 21 and 22), which is a compromise between an

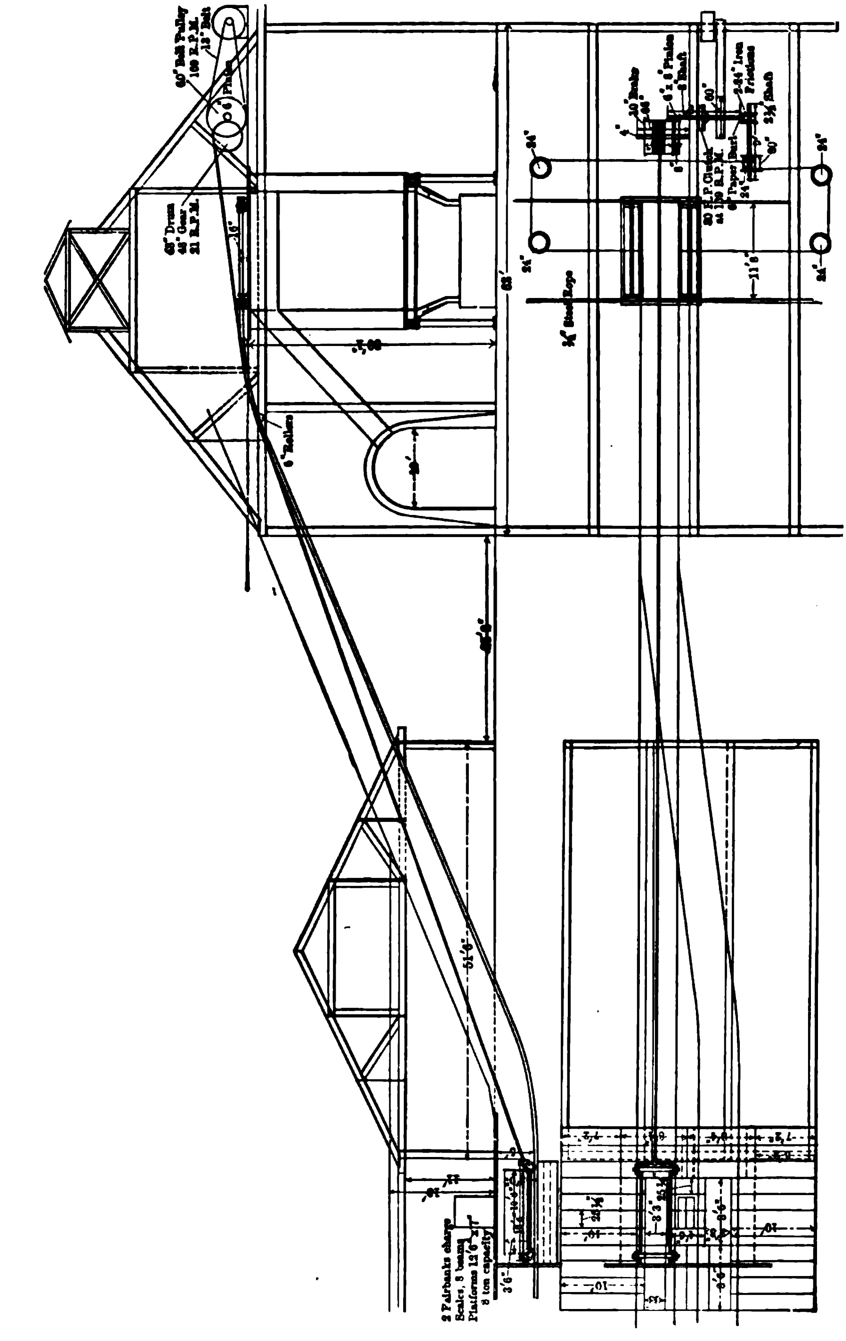


FIG. 21. — Plan of Cross Section of Automatic Feed.

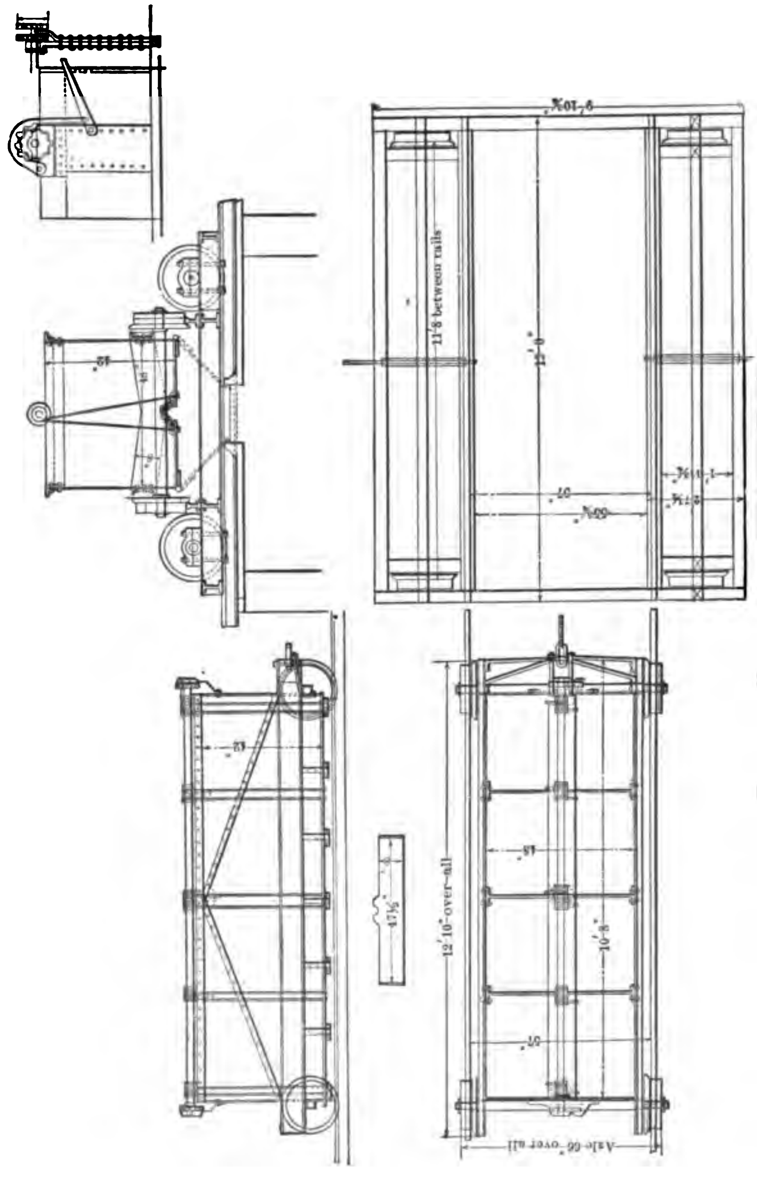


FIG. 22.—Charge Car and Transfer Truck.

automatic and a hand feed, retaining what at this date he considers the best features of both. It is a car holding 9000 pounds of ore, and with slag and fuel, a charge of 12,000 pounds. This

car is hauled up an incline to the furnace floor, then is transferred to a truck which runs on a track at right angles to the incline, and which passes over the line of furnaces. The car, having been charged below, is hauled up the incline, the cable is unhooked and the transfer truck is run over the furnace to be charged. The bottom doors are released by unwinding the windless to the shaft to which the doors are attached, and the charge is dumped while pushing the car over the furnace; about half the charge goes directly into the furnace, while half remains on

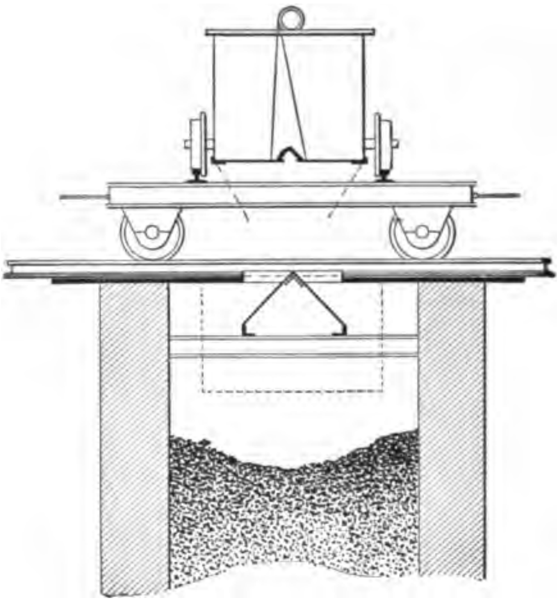


FIG. 23.

the feed floor. It was found that if the whole charge were dumped directly into the furnace, overfiring occurred, the result of the coarse pieces running to the furnace walls and thereby reducing the resistance to the blast. The half remaining on the floor is finer, and by feeding this to the walls of the furnace by hand, the overfire is kept down and the furnaces run fully as well as if fed by hand. This car feeds four furnaces 48 x 136 inches, has a capacity for charging 15,000 tons a month, and in six months saves the cost of construction in labor.

Later Experience

The central spreader shown in the charge car was too small and, on the advice of Mr. A. S. Dwight, a larger one was put on top of the furnace. (See Fig. 23.) This larger spreader caused all the charge to be thrown against the furnace walls. The finest pieces in any such mixture remain where they fall and the coarser pieces roll to the side of the pile. The furnace wall prevented the coarse from rolling in any direction except towards the center of the furnace. This gave the proper distribution of material for the lead furnace, and after its introduction there was no further trouble with the feeding of the furnaces by the car. The original car contained all the essential features, but the central spreader was too narrow and did not throw the charge to the proper place.

All the successful feeding systems embody this spreader either in the car or on top of the furnace. The Pueblo system used a deflector consisting of two leaves controlled by a lever, and the Granby system has the side-dumping car. Other reasons have been given for the successful outcome of the change, but it seems quite obvious that the above explanation is correct.

Bosh Water-jackets. — The rapid driving to increase the tonnage of the furnace, now so common, has elevated the zone of fusion, while the jackets have not been made higher, resulting in burning out the upper brickwork. Water-jackets are now being introduced for boshes in substitution for the brickwork. Fig. 24 shows such a bosh jacket.

The distance between the tap-hole and the tuyeres is frequently inadequate in rapidly driven furnaces. Where 10 inches to 12 inches formerly answered, 14 or 15 inches should now be allowed to prevent slagging of the tuyeres.

The hight necessary for the jackets to extend above the tuyeres is dependent upon how the furnace is to be run. If it is to be run on high blast and crowded to its greatest capacity, then the jackets should extend to the top of the charge, say 10 feet above the tuyere centers; but if there is to be low blast and easy running, as in lead smelting, then they do not need to be so high, since a shaft of fire-brick will serve quite as well.

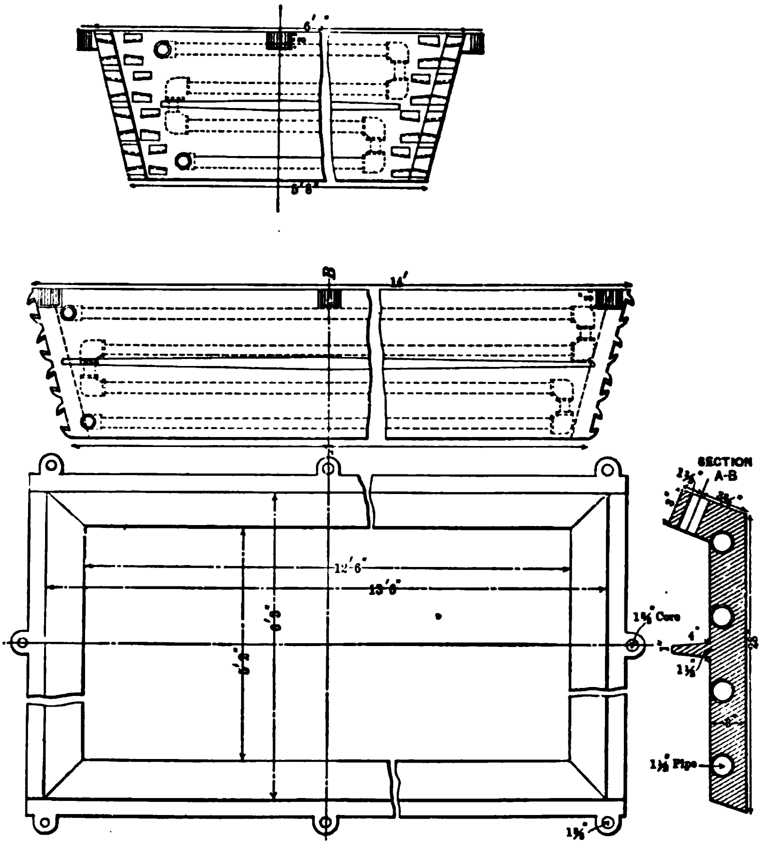
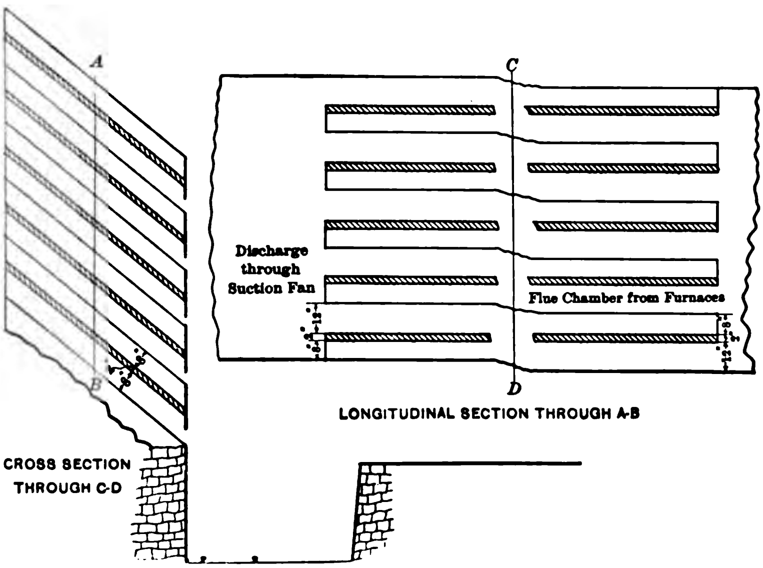


FIG. 24. — Bosh Water-jackets.

Settling. — A very marked increase has been made in the size of settlers. Formerly overflow pots were used almost exclusively, but the increase of tonnage in furnaces has made it possible to employ large boxes made of cast-iron plates on wheels, the entire box being lined with common red brick and serving as a separator for slag and matte. The size of settlers adapted to certain furnaces and slag compositions has to be worked out according to conditions. At East Helena the settler for a lead furnace 48 x 136 inches is 3 feet 6 inches by 3 feet by 6 feet inside. The behavior of this settler with slags of varying composition is peculiar. With a slag which is very clean, that is to say, free from zinc and magnesia, and containing about 34 per

cent. FeO plus MnO, and 32 per cent. SiO_2, with a fair tonnage going through the furnace, I have had them remain in use for six weeks; while on the other hand, when the slag contained as high as 15 per cent. ZnO, the settlers were closed up in two or three days. This was due to the rapid solidification of slags containing high percentages of zinc oxide. It is useless to attempt to cut the settling-pot accretions out by varying the charge. It is necessary to substitute a clean settler.

Proposed Method of Saving Flue Dust. — Higher blast-pressure has resulted, in some works, in lengthening the flues to a great



FIGS. 25 and 26. — Device for Saving Flue Dust. American Smelting and Refining Co., East Helena, Mont. Designed by H. W. Hixon, Aug. 15, 1899.

extent, while others prefer bag houses or some mechanical means of filtering the fumes in order to avoid excessive losses. Figs. 25 and 26 show a device for saving flue dust. It is a series of compartments, one above the other, so arranged that the fumes from the flue are drawn into every alternate one, and filtered through mineral wool or sized sand held between wire screens. The shelves are inclined to such a degree that the flue dust will slide out of the side of the compartment below the screens,

which may be cleaned by inserting a wire brush and brushing the bottom of the filter while in operation.

The device is connected on one end to the flue, and on the other end every alternate chamber above the filters is connected with an exhaust fan which draws the fumes through filters. Experimenting is now going on in this matter with the hope of being able to avoid using bags of combustible material. Slag wool can be made at the works by blowing steam against a stream of slag as it goes into the pots. I have found that enough slag wool can be produced in one week, by one jet of steam of 80-pounds pressure through a 1-inch pipe, to fill one of these filters.

Briquetting Flue Dust. — Briquetting of flue dust by Chisholm, Boyd & White's presses has been practised in many works. The experience gained by the operation of these briquetting machines is that they are rather expensive in repairs and cost of operation. The briquetting of fines is seemingly simple and desirable, but it depends so much for its success on the weather that at times it seems to be quite useless. If the weather is fair, either warm or very cold, briquetting is productive of good results, but if the atmosphere is humid, and the weather is either snowy or rainy, the briquettes, without drying, become soft and disintegrate very readily, producing about as much fines with very little handling as if they had not been briquetted. If frozen, they act equally as well as if thoroughly dried. I have been unable to find that there is any bad effect from charging frozen briquettes.

Briquetting Sulphides. — Briquetting fine sulphides preparatory to kiln roasting has been carried on with considerable success. In the absence of a better means of roasting iron pyrites concentrates this can be done. The briquettes are made with lime as a bond and should be dry and hard or frozen when charged into the kilns. The fuel used to ignite the sulphur is wood and fine coke. The time for roasting is about two weeks, and the amount of sulphur remaining is from 5 per cent. to 7 per cent., depending on the material to be roasted, the condition of the briquettes, and the quality of the wood used. The cost is less than calciners, and greater than Brückner furnaces.

Determination of Flue-Dust Loss. — Figure 27 is an apparatus to determine the flue-dust losses from roasting and blast-fur-

naces. A carboy is filled with water and connected with a wash bottle by means of a rubber tube. The wash bottle is connected also to a cloth filter, which, in turn, is connected with the flue by a gas pipe. When the siphon attached to the carboy is put into operation, flue gases are drawn in to replace the water. After passing through the cotton-cloth filter, nothing of value, either in silver or lead, is left to be deposited in the water in

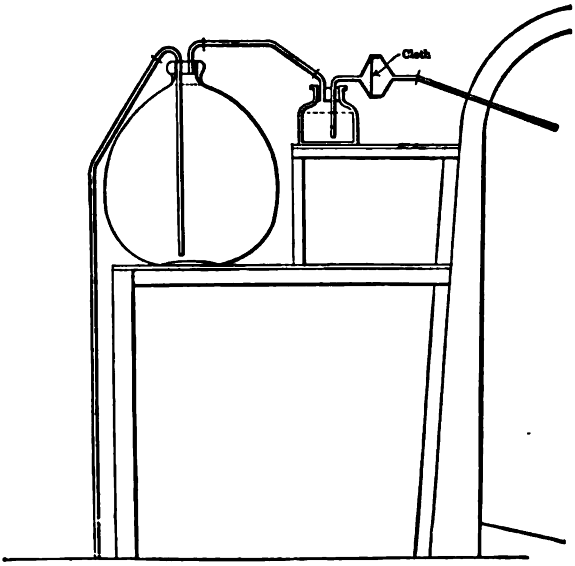


FIG. 27.—Apparatus for Determination of Flue Dust Losses.

the wash bottle. The capacity of the carboy being known and a record kept of the number of times it was filled with water and emptied, the weight of flue dust recovered from pipes and filter shows the amount of flue dust contained. The speed of the gases in the flue is determined by an anemometer, and then, by multiplying by the cross-section of the flue, the total discharge is calculated. From this the total flue-dust loss is determined by proportion as follows:

Gas filtered is to total discharge as flue dust recovered is to total loss.

XI

SMELTING RAW CONCENTRATES WITH HOT BLAST AT ANACONDA

The Experiment of smelting raw concentrates with hot blast was tried, and while the results were not satisfactory they were nevertheless instructive. Two brick stoves, similar in construction to iron-furnace stoves, were erected by using sections of old Brückner furnaces, which were bolted together to form the shell. They were built 45 feet high, 8 feet in diameter, and were lined with a checkerwork of fire-brick made in Anaconda.

The fuel used was slack coal in Taylor gas producers. The gas was burned in one stove while the blast was going through the other. The blast could he heated to about 500° F. A furnace was run for some time on a charge of 1000 pounds raw concentrates and 1000 pounds converter slag. The matte produced contained about 35 per cent. Cu (too low for converting at Anaconda), the concentration effected being about three into one.

In the resmelting of the matte with hot blast, together with converter slag, the grade was only increased to 45 per cent. Cu, showing that after the copper has reached a certain point hot blast fails to cause the elimination of iron. About 5 per cent. fuel was used during the experiments, but later this was increased to eight and hot blast was used in smelting the refuse from converting. It effected a saving of fuel as against another furnace run on cold blast, until by the alternate expanding and contracting the brick lining of the stoves gave out and the experiment was abandoned. The saving by the use of hot blast was about $25 per day, but the stove linings were too expensive to admit of any economy even at that rate.

Later Experience: Pyritic Smelting

[Republished from *Engineering and Mining Journal.*]

Pyritic smelting has been the subject of much discussion, and under certain peculiar conditions, which are purely local, it

is entitled to be called a process, as distinguished from ordinary blast-furnace smelting.

All blast-furnace smelting, even lead smelting with the use of 14 per cent. of coke, will eliminate some sulphur from the charge, which passes off with the flue gases; and some iron must be liberated and go into the slag as a result. Not more than 70 per cent. of the sulphur on a lead charge can be accounted for by the matte produced under such conditions that a heavy reducing action is indicated by the production of a small amount of speiss together with the lead-copper matte. This indicates that, even in the presence of so much as 14 per cent. coke, sulphides do burn, and, once combined with oxygen, they are separated by the reducing action of the fuel, or the gases of partial combustion.

The proper conditions for the greatest amount of *reduction*, or the reverse, the greatest amount of *oxidation*, in a blast-furnace are so much affected by the method of feeding and the arrangement of the particles in the furnace, that, without taking these conditions into consideration along with the volume of blast per minute, there is no certainty what kind of a slag will result. For example, suppose a furnace is running on a charge producing 30 per cent. copper matte, and making a slag containing 30 per cent. SiO_2 and 0.3 per cent. Cu. The percentage of fuel may be anything from, say, 5 to 12, depending upon the amount of pyrite on the charge, and also the amount of copper and the degree or absence of roast. Now, suppose we speed up the blowers or put on more blast in any suitable way; we would notice the following results: (1) The tonnage smelted will be increased, (2) the matte will contain a higher percentage of copper, and (3) the slag will contain more iron, less silica and more copper, not in exact proportion as the blast is increased, but governed by it. All of these conditions are changed by simply increasing the blast volume per minute.

Pressure does not mean anything except resistance, and it may be caused by charge burden or slag in the tuyeres, or it may mean that the tuyeres are too large or too small. After these conditions are adjusted to an equilibrium, suppose we change the method of feeding from an even distribution of fine and coarse particles to placing the coarse pieces near the wall and the fine in the center of the shaft. The result will be a further oxidation of sulphides, resulting in a further increase of iron in the slag and

a corresponding decrease of silica. The percentage of copper will increase in both the slag and matte, and the amount of matte produced will decrease. All of these changes of composition of the furnace products will be affected, *not* by any change of charge or fuel, but by increasing the volume of blast and altering the arrangement of the particles composing the charge. The furnace will not continue to run for long without crusts forming on the walls, and this may increase at one end or on one side, and diminish or entirely disappear on the other. The passage of the furnace gases is so restricted that it results in blow-holes; and these affect the oxidation to such an extent that it frequently happens that two furnaces running on the same charge will produce slags varying 3 or 4 per cent. SiO_2 or Fe, and mattes varying as much as 10 per cent. in copper content.

In this manner we may vary the action of the furnace, and if we continue to increase the blast we finally arrive at a point where the losses of copper in the slag, together with the precious metals or nickel, if there be any, would be too great to allow a further concentration on a commercial basis. The slag losses increase much faster than the grade of the matte, and while we may have increased the copper in the matte from 30 to 40 per cent. the slag will have increased from 0.3 to 0.6 per cent. or more. All this goes to show that the pyrite is not the only thing that is oxidized; copper and nickel and all the associated metals in the charge get their share of oxidation, and that is opposed to the collecting action of the matte. Slags formed under such conditions resemble slags made in the copper converter; they are foul, and contain a considerable percentage of peroxidized iron, which renders them magnetic. They will not decompose with acids without fusion with an alkali carbonate.

The remarkable effect of reversing the distribution of the pieces composing the charge is shown in lead smelting, where it has been demonstrated that it is absolutely necessary to feed the fines to the walls and the coarse to the center, in order to get the reduction necessary to prevent excessive oxidation and loss. About the year 1883 the flat-top lead furnace was introduced in nearly all the Colorado smelters, and at first they were fed through a narrow thimble; but in every case this had to be abandoned because it had the effect of placing the entire charge in the center until it sank below the end of the thimble, releasing the coarse

pieces, which rolled to the sides and ends of the shaft, while the fines all settled down in the center. The blast naturally passed up through the coarse pieces and caused overfire behind the thimble, the reduction stopped, and almost immediately the bullion production decreased to an alarming extent. These were "pyrite smelting" conditions applied to lead smelting, and were the reverse of what should have been.

The feeding was then done by shoveling the coarse to the center and the fines to the walls, forcing the carbon monoxide formed at the tuyeres to penetrate the charge to the center and act as a reducing agent all the way up through the charge. Later, when automatic feeding from a car was introduced, it was found that the charge had to be dumped so that it would be highest against the walls, so the large pieces could roll to the center, preserving the conditions as described. If at any time a number of charges were dumped in the center the conditions would become reversed, overfire would start, and bullion production would stop or decrease.

The bell-and-hopper feed of the iron blast-furnace is admirably suited for a round furnace, as it forces the stock to the outside and leaves the center lower, so that the pieces arrange themselves in the order required for reduction; but this method of feed is not suited for rectangular furnaces for lead smelting, and is not required for copper smelting.

In pyrite smelting, dumping the charge-cars from the side results in the coarse pieces going to the opposite side and the fines landing in the center, especially if the surface of the charge is 5 or 6 feet below the level of the feed door. The result is that there is a ring of fire and escaping gases around the edges of the charge, and the center is perfectly dead. I saw this strikingly illustrated during a visit, a few years ago at the works of the Tennessee Copper Company, where a man fell into a furnace that was smelting 600 tons per day. He picked himself up and walked around on the dead center and tried to climb out, until the blast was shut off and a ladder lowered to him, when he climbed out within reach of men at the feed door. The only injuries he sustained were to his hands and face and parts of his body where his clothing had burned. It was thought that his lungs would be injured, but this was not the case. Afterward it was found, by putting in pieces of paper, that there was

a down draft of air in the center induced by the blast around the sides. As the carbon monoxide did not penetrate to the center of the charge it could not exert any reducing action, and therefore the conditions for oxidation and concentration of matte were present and active, resulting in making a 40 per cent. matte out of the 3 per cent. ore.

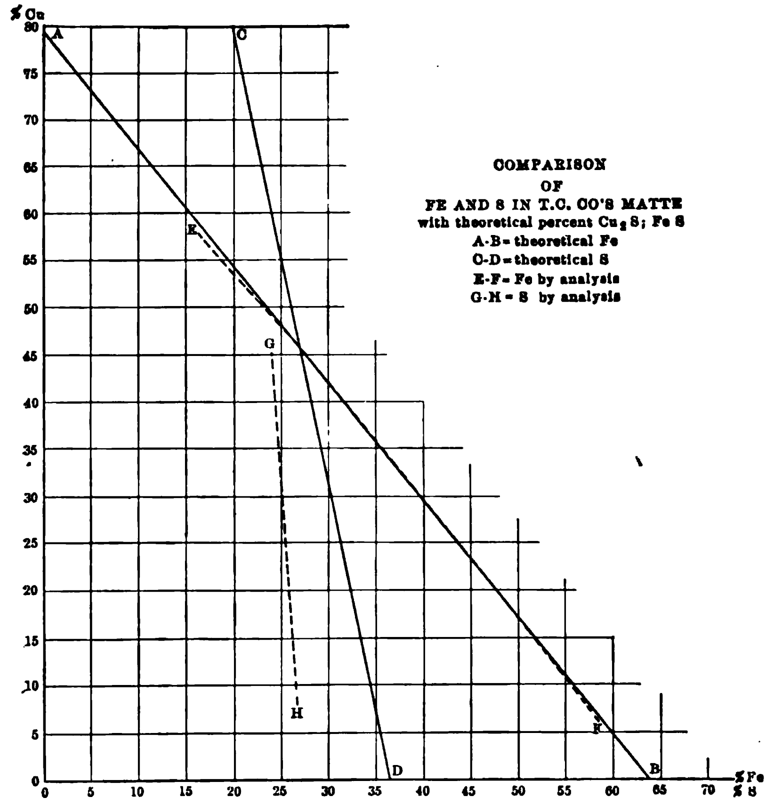


FIG. 28.

Under all such conditions of high concentration by oxidation, the resulting slags are much higher than they would be if the conditions of blast, fuel, and feeding were reversed to produce the conditions necessary for reduction. The commercial aspect of the case is, however, governed by local conditions, and depends entirely upon which method gives the greatest net return from the ore.

The two-stage operation of first making a low-grade matte 12 to 20 per cent., resmelting and concentrating this to 35 or 40 per cent., and re-smelting the slag from the matte concentration when it is high enough to justify it, seems to be the best plan devised. Trying to make a converting grade in one operation results in too high slag losses and short furnace campaigns. At the works of the Tezuitlan Copper Co., a middle course was taken. A 6 per cent. ore containing 11 per cent. Zn was part roasted in stalls and piles and smelted with an equal part of green "fine" and 6.5 per cent. coke. The resulting matte, about 35 per cent., was converted and the converter-slag added to the charge. The blast-slag contained 10 per cent. Zn and 0.6 per cent Cu. The high copper content of slag was due to high blast and zinc, otherwise the zinc gave no trouble. An ore containing zinc will make a higher concentration than if the zinc were replaced by iron.

The degree of concentration is all dependent on the blast, the fuel, feeding, and the amount of iron in the ore. An ore containing a small percentage of iron will make a higher concentration — that is, more tons of ore into one ton of matte — than an ore containing a higher percentage of iron. Hot blast may be a benefit, but it is a luxury that at many plants has been abandoned.

It is a difficult point to define, but I should say that pyritic smelting does not begin until the slag shows the characteristic peroxidation of the iron as evidenced by chilled samples refusing to decompose without fusion. If a chilled sample will not give white silica without fusion, you may be quite sure that it has been produced under conditions of oxidation characteristic of pyritic smelting.

A blast-furnace built of magnesite or chrome brick, with a steel shell of ¼-inch plate and a six-inch air-space between the steel shell and the brick, for circulation of the blast, with water-cooled tuyere points, would be an ideal construction for pyritic smelting.

The blast would be heated by the radiation of the furnace walls instead of the heat being lost in jacket-water. The water-cooled tuyeres would hold the furnace to a fixed width, and the furnace walls, braced from the steel shell and built of sufficient thickness and far enough apart, would make a strong con-

struction that would conserve the heat of the fuel and promote oxidation.

About one-sixth of the heat in the fuel is discharged in jacket-water, and to save this and return it in the blast, would greatly reduce the cost of operations, as well as prevent the necessity for so much fuel in the furnace. The presence of fuel prevents oxidation of the charge, and, therefore, a brick furnace constructed as above should make a better oxidation than a water-jacketed furnace.

Negative Results in Pyritic Smelting[1]

Under the head of "Negative Results in Pyritic Smelting," Mr. G. F. Beardsley described his experience at Copper Cliff, Ontario, in attempting to adapt to the copper nickel ore of the Sudbury district, the practice of raw smelting by which the Mt. Lyall copper ores are so successfully treated. Mr. Beardsley's Mt. Lyall experience especially fitted him to attempt what so many others have failed in; and the result of his experience is therefore especially interesting.

Referring to his article in the *Journal* of August 24, I wish to give the result of similar experiments in the smelting of raw copper nickel pyrrhotites, and the reasons for failure, which differ from those assigned by Mr. Beardsley.

In all of the experiments described by Mr. Beardsley, the furnace was running on a charge of roasted ore with 13 per cent. of coke and this was changed to a charge of raw ore with 3½ to 6 per cent. of coke. Mr. Beardsley states that the tests lasted only from two to four hours, and the difficulties began to show within an hour after the change of charge. This is similar to my own experience, but I take an entirely different view of the cause of failure to effect concentration by oxidation of the sulphides. The 13 per cent. of fuel on the charge that preceded the raw charge was not entirely consumed when the raw charge got down to the smelting zone in the furnace, and the heat and reducing action of this remnant of fuel was sufficient to melt part of the sulphides and run them out as low-grade matte with the results as described.

If the furnace had been started on 3½ per cent. of fuel instead

[1] Published in the *Engineering and Mining Journal*, September, 1907.

of following a charge with 13 per cent. it would not have started with the sudden rush of low-grade matte as described; probably it would not have started at all. I have tried this variation and found that the furnace was, so to speak, left at the post. This indicates to me that more than 3½ to 6 per cent. of coke is necessary to smelt a charge of this character. In other words, the fusing point of the ore is higher than a charge of ore containing only copper, which would run on the percentage of fuel that a charge of nickel copper ore would not run on. The fact that the furnace almost stopped smelting and was rescued with difficulty as soon as the remnant of the 13 per cent. of fuel had burned out, proves that this is the true cause, and not, as Mr. Beardsley states, the lower fusing point of the sulphides which he says liquated out, and left the furnace full of a skeleton of rock matter.

To illustrate the effect of fuel already in the furnace, I recall an annoying experience that happened in East Helena. An order had been given to the feeder on a furnace to reduce the amount of coke on the charge. He misunderstood the order and did not put in any fuel, and several hours later it became apparent that something was amiss. Inquiry showed that about 10 tons of charge had been fed without fuel. Immediately an extra amount of fuel and matte was fed in, the blast was increased, and the furnace pulled through without a freeze-up. This happened on a lead furnace where about 15 per cent. of coke was being used.

According to the analyses which Mr. Beardsley gives there is less than 20 per cent. of insoluble or rock matter in the ore, and if the furnace practically stopped smelting within such a short time as he describes, it is impossible that so much rock matter could accumulate in the shaft of the furnace.

I do not find anything to criticise in Mr. Beardsley's statement of facts or the results, for I have been all over the same road, but I think the conclusions he draws from them are erroneous. The conclusion that I have come to is that the fusing point of the ore is higher than a similar ore containing copper only. The charge must be fused or it will not smelt. If the fuel necessary to melt it is put on, it prevents oxidation; and without oxidation, concentration is impossible. Take off the fuel and the furnace stops; put it on and it prevents concentration.

As a compromise between the methods, we smelt pile-roasted ore with green ore. If we could smelt the ores raw and produce

a converting grade of matte, the slag losses would probably be too high for commercial practice. The slag will carry about as much nickel as copper, irrespective of the fact that, if only copper were present, the metal loss would be one-half as much as the sum of the two. Smelting with a high degree of oxidation makes fouler slags on the same grade of matte than would be produced with more fuel and reduction.

In the process of smelting and converting copper ores and mattes the reaction between the copper oxide and the sulphide is exothermic and adds energy to the process. The same cannot be said of the reaction between nickel oxide and nickel sulphide, and it is for this very reason that pyritic smelting as described by Mr. Beardsley was not a success on nickel ores.

XII

COPPER CONVERTING AT ANACONDA

AT THE time the writer entered the employ of the Anaconda company for the purpose of constructing a converter-plant to treat the entire tonnage of matte produced, about 300 tons daily, there was in operation a converter-plant of twelve vessels that had been erected as an experiment. The workings of this plant had been very unsatisfactory, and the short life of the linings, as well as the large losses of copper had caused it to be severely criticised. At that time the process of copper converting was not very well understood, and it was hoped to find a lining that would last a week, as in the case of steel-making. That has proved the great stumbling-block for many, though the difference in the processes is apparent. In steel-making the charge is blown only long enough to burn out the excess of carbon, and before the iron has begun to oxidize the converter is turned down and the charge poured out. All of the products of combustion in this case are gaseous except what SiO_2 may be formed by the silicon present in the cast iron. This SiO_2 has a protecting influence on the lining of the vessel, since it will combine with any oxide of iron formed and supply SiO_2 that would otherwise have to come out of the lining. In copper converting the matte may contain from 13 to 35 per cent. of Fe in the form of sulphide, all of which has to be converted into the oxide before it can be separated from the copper and sulphur. The 13 per cent. of iron would indicate a matte of 60 per cent. Cu, and the 35 per cent. of iron about 28 per cent. Cu, these being the limits within which the writer has converted.

In the case of a 60 per cent. matte with 13 per cent. Fe, each ton of matte charged into the converter would contain 260 pounds of Fe. A partial analysis of the slag formed after the charge has been blown to the skimming point is SiO_2 37, Fe 38, Cu 3, showing that for each pound of iron in the matte a pound of silica has to be provided in order to form a slag which will, with the

limited heat generated by the combustion of the Fe to FeO and the partial combustion of the sulphur to SO_2, remain fluid and admit of being poured out of the vessel. It must be borne in mind that it is not only a question of oxidizing the iron, but of separating it from the charge after it has been oxidized and removing it from the converter as quickly as possible. This can only be done by forming a fusible slag, and a slag can only be formed by the union of an acid and bases, and as the bases are formed naturally in the converter the acid must be present to unite with them as fast as they are formed, otherwise there will be an accumulation of very infusible FeO in the converter which will mix up mechanically with the white metal (sulphide of copper), and make a spongy, viscid mass which cannot be skimmed or finished to copper. The oxide of ircn alone without silica will not form a fluid slag, so that if it were possible to run a charge in a water-jacketed converter without freezing, which it is not, it would still be impossible to remove the FeO from the converter because it would not separate from the sulphide of copper.

All chemical unions are attended with the development of more or less heat, and it is quite probable that the union of FeO with SiO_2 to form a slag, is attended with a considerable heat development; and if the converter were robbed of this by the use of any other kind of lining the heat would be insufficient. As for basic lining and water-jacketed converters, the writer has tried them both and will give in detail the results so that aside from theory the actual outcome may be known.

The first experiment to be tried was to protect the lining immediately above the tuyeres where the corrosion is greatest. This was attempted by imbedding in the silica and clay lining a pipe coil running around three sides of the converter and immediately above the tuyeres. This coil was of 1¼-inch pipe, put together with malleable ells and clamped to the shell of the vessel by stay-bolts about 8 inches in length. The coil was connected at the inlet by a hose to the water main in which the pressure was 45 pounds; the discharge was also provided with a hose connection, so that the converter could be turned up or down without interfering with either. The lining was put in in the usual manner and thoroughly rammed behind the pipe coil, and as the lining was about 20 inches thick at the tuyeres the charge could not come into contact with the coil until the 12 inches of clay cover-

ing had been eaten away. The first and second charges were run as usual with the rapid decrease in the thickness of the lining and increase in size of the vessel. The third charge cut away all the lining in front of the coil, and when the converter was turned down to skim slag, the pipes could be seen bare and exposed from the nose of the converter. At this point the experiment was attended with extreme danger, for if the pipe had given way, or had been attacked by the matte, an explosion would certainly have occurred that would have wrecked the entire plant as well as killed the men working on the vessel. For this reason everybody kept at a respectful distance while the converter was blowing, and only approached when necessary to turn the vessel up or down. But as nothing occurred to frighten the men, they gradually came to regard it as safe, and returned to work as usual.

The charge was finished and the copper poured, when it was found that large lumps of copper were adhering to the tuyeres and pipe, but at other points the lining had been corroded as much as usual. A fourth charge was attempted, but the bottom of the vessel became too thin, and the charge was lost on account of breaking through at that point.

It was thus demonstrated that protecting the lining at one point only changed the corrosion to another, so that it became a question of abandoning the experiment or putting in pipe enough to water-jacket the whole interior. This was done, and a coil was constructed and put into the converter which should be imbedded in and should protect the lining as high as the top of the charge. The converter was charged and the first charge finished without exposing the pipe; the second charge was blown to slag and skimmed, but ran cold on the finish blow, and more matte was tapped in and blown to slag and skimmed again. By this time the pipes were exposed throughout the interior of the vessel and the lining had been eaten away from behind them in some places. The attempt to finish the charge was a failure, and it had to be poured out as white metal and used as scrap in the other converters. Many other attempts were made with this converter, jacketed as it was, but they all ended in failure and demonstrated beyond a doubt that a charge could not be blown to slag after the pipe became exposed, for the reason that there was no silica to flux the basic FeO formed by the oxidation of the iron in the matte.

Water-jackets were then abandoned and a lining composed of burned lime mixed with coal tar was tried. The result was a foregone conclusion, but nevertheless we were hunting for straws to grasp and it was given a fair and impartial trial. Just enough tar was used to stick the lime together, and when in place the lining was baked with a coke fire to drive off the gaseous portion of the tar. A charge was run into the vessel and blown until the flame at the nose indicated the complete oxidation of the iron. The converter was then turned down to skim, but in place of fluid slag and matte there was found an indescribable mush of matte, lining, and oxide of iron, with which nothing could be done, and after one more trial with the same result the experiment was abandoned.

In conclusion, and before leaving the subject, it is well to state that the silica lining for a copper converter serves a double purpose, that of a lining and also a flux, and that it is necessary to the success of the process that the lining should be corroded. If this corrosion were stopped by any means except the introduction of silica in some other way, the process would be defeated. The attempt to introduce silica through the tuyeres with the blast has thus far proven a failure, and very likely always will, for mechanical reasons. In the first place the time during which SiO_2 is needed is very short, and the quantity required is too large to admit of even a small part of it being introduced.

Take the case of a five-ton charge of matte containing Cu 35 per cent., Fe 30 per cent., S 26 per cent. Each ton of matte would contain 600 pounds of Fe, and the five tons would require 1620 pounds of SiO_2 to be introduced through the tuyeres in the space of one hour in order to make a slag with 25 per cent. SiO_2, and 60 per cent, FeO.

The lower grade in copper the matte is, and the higher in iron, the more basic will be the resulting slag, and while 55 to 60 per cent. Cu matte will make slags as high as 40 per cent. SiO_2, a 35 per cent. Cu matte will make a slag with about 25 per cent. SiO_2, and 60 per cent. FeO.

The rapid corrosion has the effect of making the lining lose its binding force and it falls off in chunks, just as the rapid erosion of the banks of a stream by water will make them cave off in large pieces. Consequently the life of a lining is not exactly inversely proportionate to the amount of iron in the matte, but

decreases in length more rapidly than the iron increases. For example, a lining that would produce 200 bars on 55 per cent. Cu matte with an iron contents of 17 per cent. would probably not produce more than 50 on a 28 per cent. matte with the Fe 35 per cent. But there are so many chances for weak points to develop unexpectedly in the lining with the low-grade mattes that no comparison is of much value.

After testing all the different schemes that could afford a possible solution of the difficulties of relining converters in the stands, it became apparent that the proposition was a mechanical one; to handle the converters as quickly as possible and to make a change of vessels in the least possible time when a lining was destroyed. The practice at the old plant at Anaconda, and for some time at the Parrot, was to run the converter until the lining became too thin to stand another charge, and then to flood the vessel with water from a 3-inch hose and allow the stream to run until the temperature had been reduced so that after the water had been turned out a man could go inside, cut out all the loose jagged points and slag shell, and then reline the vessel. The cutting out was done while the vessel was turned bottom up, and the loose material was thoroughly cleaned out before any more lining was put in. The converter was then righted and the lining passed in with a shovel through the nose in lumps, about 8-inch cube. The liner first put in the bottom by throwing the lumps of lining against the bottom of the converter and afterwards pressing them in place with his foot. The bottom should be from 4 to 6 inches below the tuyeres and made as thick as the side lining. The bottom finished, a circular board about the size of a barrel-head was put on top of the clay for the liner to stand on, and with this as a form the side lining was built up against the old lining remaining in place, or against the shell of the converter if it had all fallen out in the cutting-out process.

Large mitts were used by the liners, and a trowel to cut off and shape the lining after the lumps were pounded into position with the fist. Attempts to use all kinds of tamping-irons were made, but it was found that it took much longer to put in a lining with them than with the hands, and there was no apparent improvement in its character.

The lining was made of pure white quartz crushed to the size of slack coal in crushers and rolls, and afterwards ground

in a Chilian mill with one shovel of fat, sticky clay to 8, 9, or 10 shovels of quartz, according as the clay seemed to vary in plastic or binding qualities. The clay employed contained a somewhat large percentage of alkali earths as well as iron, and only so much of it was used as was necessary to stick the quartz together when moistened and ground in the Chilian mill. On

CLAY ANALYSIS

SiO_2	66.0 per cent.
Al_2O_3	18.5
Fe	3.1
CaO	2.9
H_2O	8.4
	98.9

account of the flooding of the converter before relining, very little difficulty was experienced in making the new lining adhere to the old, but later, when the new plant was constructed, and the converters were relined without filling them with water, it would sometimes happen that the fresh lining would part from the old, and linings put into a dry vessel did not last as long as those put into one that had been made thoroughly wet. Notwithstanding this small advantage, the use of water for cooling the vessels is a poor policy for many reasons; the first of which is that there are formed by the blowing of the charge, sulphates of iron and copper, which are dissolved by the water and react on the ironwork of the converter, corroding it rapidly and causing it to break open at the riveted seams after a couple of months continuous run. The lining remaining in the vessel swells by the addition of water and brings a heavy strain on the converter, frequently causing splits of the seams. Moreover all the copper that is dissolved as sulphate is carried away and is lost unless recovered on scrap iron.

At this plant, which, as stated, was an experimental one, there were three converters to each cupola or melting furnace, and the charge was run from the cupola to the converters in long spouts. One of the three converters was kept in operation while one was drying out and the third was relining. By putting on extra crews when the matte was of good grade, it was sometimes possible to work five or six of the twelve vessels at one time. But this was only possible when the copper in the matte was as

much as 60 per cent. and the iron as low as 13 per cent. Otherwise the linings would be destroyed too rapidly to admit of running more than one vessel out of the three. Owing to the bad dust-chamber arrangements, and to the use of water in the converters, the losses at this plant were exceedingly high. During the year and a half that the writer ran it, before the new plant could be constructed, the losses in copper were 4 per cent. and silver 5 per cent., while the cost of converting varied between 0.72 cents and 1.34 cents per pound of copper converted from matte averaging 55 per cent. copper.

The converters were 60 × 60 inches, square in section, and turned by a worm gear operated by power from a line shaft. The plant was put into a building which it did not fit, or rather the building did not fit the plant, and, generally speaking, everything worked at a disadvantage. The experience gained was the basis for constructing the new plant and was valuable from that point of view. The largest production in any single month from this plant was 5,500,000 pounds of copper.

Previous to constructing the new plant several systems of handling converters were developed, and their merits discussed. It was finally decided that the handling by crane was preferable to the car, and the detail plans were drawn up accordingly.

It was unfortunate, though unavoidable, that the plant had to be built with remelting furnaces, the smelting works being already established and so compact in arrangement that it would have been impossible to take the matte from the reverberatories to the converters in a molten condition. This is a point that in any new construction should be kept in mind, even though converters are not to be installed at first. The extra expense of remelting matte will be about $2 per ton where coke is $12. There is a small loss by flue dust in addition, so that on 50 per cent. matte the remelting expense would be $4 per ton of copper produced, or 0.2 cent per pound of copper converted, which is approximately the difference of cost in favor of a plant where the matte is taken directly from the smelting-furnaces to the converters.

This cost would be much less in a locality where labor and fuel are cheaper than in Montana, but in most places in the West the figures would apply, and the cost per pound of copper would also increase rapidly as the grade of the matte fell below 50 per

cent., or the tonnage decreased. For instance, when the tonnage handled at the converter plant was small the items of general expense were just as much as when it was large, and would greatly affect the result for that month.

The average cost for converting 55 per cent. matte may be stated at 0.65 cent per pound of copper, divided as follows:

Remelting matte	0.2
Labor and lining for converters	0.25
Labor on converters	0.1
Resmelting converter slag	0.05
Supplies	0.05
	0.65
For a plant without remelting	0.45

The losses in converting, as shown at Anaconda, were about 3 per cent. of the copper contents, but there was a wide difference between assays of samples of matte taken at the converter plant and at the smelter. This difference amounted to 0.5 per cent. on all the matte received, and represented about 1 per cent. of the loss, so that the real loss was probably only 2 per cent. of the copper.

The silver loss was apparently high according to assays of the converter copper; but the casting showed a corresponding gain; and after making allowance for this the loss of silver in converting was reckoned at less than 1 per cent.

History of Matte Converting

[Published in *The Engineering and Mining Journal*, Aug. 4, 1906.]

The first successful experiments with the converter as applied to copper matte are credited to Pierre Manhès; but many attempts had previously been made, and several patents were issued. However, the experiments failed, because no provision was made for punching the tuyeres.

Manhès' invention consisted in placing a hole in the outer surface of the tuyere-box in alinement with the tuyeres, so that a rod (slightly smaller than the tuyere) could be driven through this hole into the tuyere, through the lining, and dislodge any obstruction at the tuyere end. In his patent specification, the word "air-belt" is used in place of tuyere-box; this gave the opportunity for placing the tuyere in a box separate from the air-

belt (to avoid infringement of the letter if not of the principle of his patent), and was quickly utilized.

In 1892 I was engaged to build a converter plant for the Anaconda Copper Mining Company, and, on the advice of the company's attorney, the converters were constructed on the principle described above. I do not believe that the point was contested.

The holes through the outer surface of the tuyere-box were closed by wooden plugs (which had to be driven in tight enough to hold against the blast-pressure of 10 to 15 pounds to the square inch. The matte thrown out of the converter during the blow would ignite the outer ends of these plugs, and the heat would contract them; so that, between the pressure from within and the fire from without, it was like standing in front of a gatling gun to punch the tuyeres. A plug weighing 2 ounces and propelled by a pressure of 15 pounds, could knock a man out completely; and adding to this the chances of having a shower of fluid matte fall on him, he certainly earned his $2.50 for a 12-hour shift.

Mr. J. A. Dyblie (who was associated with me in designing the plant) changed all this, by inventing the ball-valve which bears his name, and for which he receives a royalty from the manufacturers. These valves were so arranged that when the converter was in position, to blow, each tuyere-hole in the outer surface of the tuyere-box was automatically closed by a steel ball slightly larger than the hole. Guides were so arranged that the punch-rod would lift the ball away from its seat, and return it to its place when the rod was removed.

Since that time other valves have been introduced which embody the same principle; such as the use of a small cylinder, in place of a ball-, a flap-, or clack-valve, hung loosely at the upper ends.

The Parrot Company in Butte is said to have first used the converter on copper matte in America. The H. H. Vivian Company, operating the Murray mine near Sudbury, Ont., was the first to apply it to copper-nickel matte. The Chicago mine (operated by R. P. Travers) also used a small converter on copper-nickel matte at a later date. This mine did not continue long in operation; the same is true of the Murray mine. Their converters were of the barrel type, and not much larger (probably 5 feet long by 4 feet diameter); they turned by a screw operated by hand or by a belt from a shaft.

In the converting of copper matte, the iron is all oxidized, and, after the resulting slag has been poured off, the remaining high-grade matte is blown up to blister copper and poured into bar molds. This cannot be done with copper-nickel mattes, as the nickel begins to oxidize and remains in that condition after the iron is removed. It is therefore necessary to stop the process at this point and to pour the matte into molds, when the iron has been reduced to 1 per cent. or less. For these reasons, all the converting plants which have operated on copper-nickel ores have produced only high-grade mattes carrying copper and nickel in varying proportions, depending on the metal contents of the ores from which they were derived. The sum of the two metals is, approximately, 80 per cent., with 18 per cent. sulphur, 1 per cent. iron, and 1 per cent. oxygen.

The Canadian Copper Company used converters in a small way at a later period than the H. H. Vivian Company; but it was claimed that the loss of nickel was excessive; and, on that account, that the process could not be successfully applied to ores carrying nickel.

The plant which I constructed for the Mond Nickel Company disproved this contention; and at a later date the Canadian Copper Company introduced the latest practice of large furnaces, and large converters electrically operated; it now has the largest and best equipped plant in Canada. The electric transmission (from the high falls of the Spanish river) to Copper Cliff is over 25 miles long, and has a capacity of 5000 h. p. The converting process is now producing nearly all the copper derived from sulphide ores, and all the copper and nickel from the Sudbury district.

The practice of converting is interesting, and varies considerably in different localities. The original practice at Anaconda and Butte was to line the converters with a mixture of crushed quartz, with enough clay to bond it. The lining was put in wet, and simply plastered together by pressure of the liner's hands, feet and knees, while standing in the converter. The top and bottom sections were lined in the same manner while together in the stand.

After the introduction of the traveling crane, three or more shells were provided for each stand; the vessels were taken out to be lined, the top section removed, and the interior sprayed with water. After cutting out the slag shell, the new lining was

tamped in between the old lining and a wood or steel form. The thickness of a new lining would depend upon the diameter of the vessel; in the largest sizes, it was about 30 inches at the tuyeres, and tapered off to nothing at the mouth. The mixture for tamping was not so wet as in the other method described, but was otherwise the same. The top sections of converters may be tamped, but are generally lined by hand after being placed on the bottom section. A tamped lining frequently falls out while it is being turned over to put it in place. If the mixture is adhesive enough, it can be done; but the clay generally available will not stand such shocks, and the result of several hours' labor may be lost in placing the top on the bottom section.

At Tezuitlan, Mexico, the Pachuca *lamas*, or clay ore from the silver mines, was used for lining material. This consists of almost pure silicate of alumina, with 10 to 20 ounces of silver per ton, and is very adhesive. The tops were tamped the same as the bottom section. The joint was made of a special wet mixture of the same composition, placed on the bottom section; the top section was lowered down upon it by the traveling crane. Cutter bolts and keys drew the two sections firmly together, and the lining was ready to be dried out and heated before being placed in the blowing stand.

This drying out and heating is important, and necessary to the success of the blowing. If a lining is wet and cold, the slag, if formed at all, will be too cold to separate from the matte; the result will be a granulated mixture which will have to be poured out and broken up. As the size of the interior is small on the first charge, it is necessary to have the lining next the charge red hot although the heat may not penetrate more than 2 or 3 inches; and the remainder of the lining may be quite wet. The matte should also be quite hot; for, if cold, it will act slowly, and may produce the same result as if charged into a cold vessel.

For this reason I prefer tapping direct from the settler into the converter; but nearly all plants use a ladle and traveling crane to transfer the charges from the settler to the converters; in large plants this is necessary because of having one large furnace to supply several converters.

The use of silicious silver-, gold-, or copper-ores for lining material is one of the most important features; it can be made a source of revenue instead of an item of expense, as when barren

quartz is used. The vessels should be large enough to get the greatest thickness of lining; and deep enough to permit the use of 15 pounds of blast without blowing the charge out of the mouth of the converter. The first use of silicious ores for lining was attempted at the Parrot in Butte; but the vessels were too small and the linings gave out too rapidly when made of material less refractory than quartz and clay; the practice was abandoned.

In 1895, at the Aguas Calientes plant, we used the Pachuca *lamas* for lining the big converter. I have been informed that since then this practice has become general in Mexico.

The shape of converters has been supposed to have a decided effect on the blast pressure required; and the claim has been made that the barrel type of converter could operate on 5 pounds pressure, where the upright or round type would require 10 to 15 pounds. I have operated both types, and I fail to find any difference. The whole question hinges on the depth of matte over the tuyeres; and, as it is necessary to have the blast penetrate about 2 feet or more of a depth (in order to give time for the combination of oxygen with the sulphur and iron), the shape of the vessel does not enter into the question. It is easier to move the barrel-type converters on and off their stands; the stands are much simpler and cost less, which is probably the reason for their greater popularity.

The life of a lining varies with several factors, and may be long or short, depending upon all or any one of them:

First, the grade of the matte is important. Matte high in copper will be low in iron, and *vice versa;* and, as the iron from the matte requires silica and gets it from the lining, the more iron a matte contains, the shorter will be the life of the lining, other things being equal.

Second, the composition of the material from which the lining is made affects its resisting or refractory power. A lining made of pure quartz, crushed fine and bonded with a fat refractory clay, would at first appear to be the best lining that could be made; but I have found that this is not so. A quartzite containing 87 per cent. silica and 10 per cent. alumina, bonded with the same clay as the quartz, and converting the same grade of matte, gave results 50 per cent. better.

For example, the silica lining would last for an average of six charges on 27 per cent. matte; the quartzite lining would convert

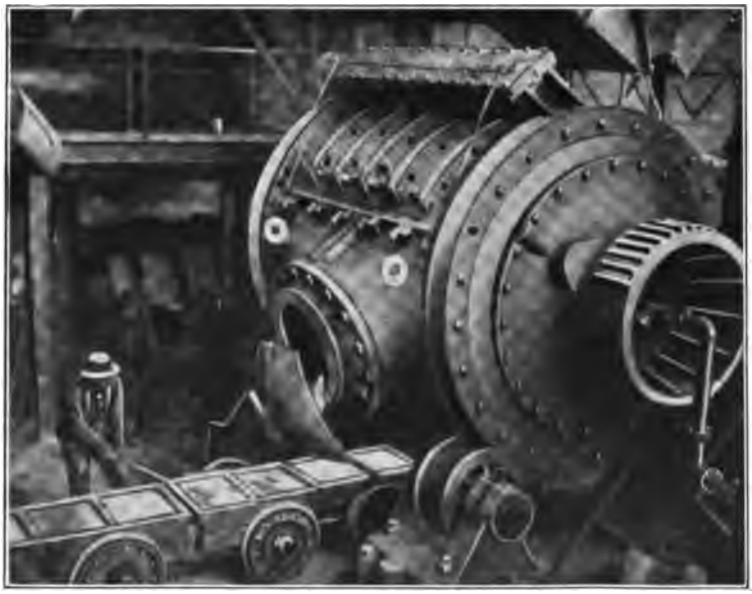


Fig. 29.

eight charges. As the last two charges contain as much matte as the first four the actual showing was rather better than 50 per cent.

Feeding silicious material into the converter while it is blowing has a beneficial effect on the life of the lining, as it supplies silica that would otherwise have to come out of the lining. The life of a lining can be increased 50 per cent. by additions of silicious material through the mouth of the converter while blowing, or by charging into the vessel before the matte. All such material should be thoroughly dried beforehand, or serious explosions are likely to result.

In order to dry this material I have found it convenient and effective to charge into the downtake from the blast-furnace, and draw it out of the flue as needed. The flue dust which is mixed with it does not interfere, as it is present only in small quantity. The lining should be compact enough to prevent the escape of too much blast through the lining without penetrating the charge, and at the same time porous enough to promote rapid drying and the escape of steam.

It has been found that a considerable portion of the blast enters the charge through small openings in the lining, other than the tuyeres. When a converter is turned down, to pour off slag, these small capillary openings may be seen as small dark spots, showing against the incandescent interior. They are dark because colder than other parts of the interior, due to the chilling effect of the blast; but they quickly become red and disappear when the blast is shut off.

The chemistry of copper matte converting, and the fact that metallic copper can be produced from copper sulphide, depend upon the reduction of the copper oxide by the sulphide. This takes place throughout the blow, and prevents the copper oxide from going into the slag in large quantity. In the latter portion of the blow (when all the iron has been removed and only copper sulphide remains), the reduction must keep pace with the oxidation, or a slag would be formed before the finish, which would be high in copper oxide.

The fact that this slag does not form until the charge is overblown proves that the reaction:

$$2\,CuO + Cu_2S = 4\,Cu + SO_2,$$

takes place as rapidly as the copper oxide is formed.

If a charge is overblown slightly before pouring the slag, the large amount of slag will absorb the copper oxide more rapidly than the reduction takes place. The result is that when the converter is turned down, changing the equilibrium of the sulphides and oxides, foaming and shooting from the mouth of the converter result. Indeed, if the overblow has gone far enough, the entire contents of the converter may be thrown out by the sudden liberation of sulphur dioxide.

The reaction between nickel sulphide and nickel oxide does not take place as rapidly as in the case of copper. The result is that in the after-blow (when all the iron has been removed) the nickel begins to oxidize; this nickel oxide combines with silica to form a thick, pasty slag, which chills on the interior of the vessel and clogs the tuyeres, preventing the blast from entering. It is difficult to blow a charge of matte containing nickel after the iron has been eliminated; and it is impossible to finish a charge to metal in the same manner as with copper alone. The nickel will be oxidized before the sulphur is all out, and the resulting metal will not contain more than a fifth of the nickel that it should when there is still 5 per cent. of sulphur remaining. This accounts for the fact that all mattes containing nickel are converted only to remove the iron; the process is stopped when the metal content reaches 80 per cent.

In 1905–06, at the experimental plant of the Pittsburgh and Montana Co., Butte, W. A. Heywood, operating the Baggaley process, used, with wonderful success, a thick steel shell converter, lined with magnesite. [See *Engineering and Mining Journal,* March 24, 1906.] The remarkable results obtained on matte carrying as low as ten per cent. of copper constitute such a radical departure from all established practice that further developments are awaited with interest. The smelting of raw sulphides has always been handicapped by the production of large quantities of low-grade matte, which could not be converted in silica-lined converters, because of the too rapid destruction of the lining.

If, as seems probable, this matte can now be converted in converters lined with magnesite, all the necessary silica being fed in before or during the blow, then the proper way to treat such ores is to smelt them raw, with the fuel necessary to make the furnace run at a good tonnage, and convert all the matte.

This will avoid the necessity for the two-stage smelting practice, at present in use at many plants; and, at the same time, it will produce blast-furnace slags much lower in copper content than previously obtained.

While the magnesite lining has proved such a success on low-grade copper mattes, to my regret I found that on nickel mattes it did not operate as well as the silica lining. The chief difficulty was in getting the blast to enter. The tuyeres became clogged and needed continual punching. If allowed to run without punching, in less than a minute the blast would be shut off entirely. The lining was put in, in the manner described by Mr. Heywood, and all the conditions were as usual with a silica lining. We succeeded in finishing about one charge in four, the others being too cold to produce fluid slag. The blast-pressure was raised from 10 to 18 pounds per square inch without effect.

I am of the opinion that the presence of silica at the tuyeres is necessary in the case of nickel matte, in order to combine with the small amount of nickel oxide formed; as otherwise it chills and stops the tuyeres.

Later Experience

Since writing the above, I have experimented further with a magnesite-lined converter on copper-nickel matte. The result was the same in each case, except that by feeding in very little silica the temperature of the charge was raised to a dazzling white heat, and under these conditions the charge was blown successfully but the magnesite lining was fused out. This indicates that the presence of a silica lining is necessary to cause the reaction between the nickel oxide and the nickel sulphide at a reasonably low temperature, or that in the absence of a silica lining the reactions in the converter require a much higher temperature; which is fatal to the magnesite lining.

I am of the opinion that the reaction between copper oxide and copper sulphide is exothermic and that the reaction between nickel oxide and nickel sulphide is endothermic, and that this difference is sufficient to cause the failure of pyritic smelting as applied to nickel ores, as well as the failure of the magnesite-lined converter on nickel mattes. There can be no doubt that Mr. Heywood made a success of the Baggaley process of converting low-grade copper mattes in Butte, and unless there is some good

chemical reason, as above suggested, it should work on nickel mattes.

It is further plain that the reactions in the copper converter must take place at a lower temperature than that at which the magnesite will combine with the slag, otherwise the lining will be destroyed, as it was in our experience with nickel matte, when we got the temperatures high enough to cause the reactions in the absence of a silicious lining.

XIII

BLOWING A CONVERTER CHARGE

THE OPERATION of blowing a charge in a converter is one requiring much experience and attention on the part of the skimmer, and can only be learned by actual practice in the work. A knowledge of the changes of flame coloration at the nose, indicating the condition of the charge, is only attained after the apprentice has given much attention to it, and in some cases where color-blindness may exist it is impossible for him to do so at all. The first portion of the blow is usually from 40 to 60 minutes' duration, although if the charge be too heavy it may be prolonged for much longer time. The flame is a light green color, with an occasional shade of yellow in the first stages, probably due to volatilized sulphur. As the time for skimming approaches occasional flashes of azure blue may be seen mingling with the light green, and if sufficiently prolonged it will become wholly azure blue. The converter must be turned down and the blast shut off before this change in the flame coloration has gone too far, or the entire contents of the converter will frequently be blown out and scattered over the building. The writer has seen charges weighing several tons foam and shoot matte thirty feet in the air from being overblown only a few minutes.

The cause of these explosions is the oxidation of a portion of the copper which enters the slag, and when the vessel is turned, or the equilibrium of the slag and matte is disturbed, it results in the mixing of slag containing oxide with the white metal below; the sulphur of the white metal has a reducing effect on the copper oxide, and the result is a sudden precipitation of copper and the formation of large volumes of SO_2 gas, which causes the charge to foam and at times throw the greater portion of it out of the converter. This may happen before the converter is turned down owing to the agitation of the charge by the blast, but it is most active while the vessel is moving. It may happen after the charge has been skimmed, but in this case it is usually not

so serious, and shows that the skimming has been done too soon, or before the iron has all been oxidized; that portion of the iron remaining in the matte having formed slag, which has absorbed copper oxide from the charge below, and has been acted upon by the sulphur in the white metal. It is frequently necessary to skim a charge twice in order to remove the slag, and take some of the burden off the blast. This will allow the blast to penetrate the charge faster and will save time, as well as avoid the possibility of any overblow. It also equalizes the charge burden in cases where several converters are furnished blast from the same pipe.

The formation of copper oxide will not take place to any great extent as long as iron is present in the matte, but as soon as the iron is all oxidized and gone into the slag as silicate, then copper begins to oxidize rapidly, and if the slag is not taken off before this begins it will unavoidably result in a large portion of the slag being thrown out. After the slag has been skimmed off, the copper will be oxidized just the same, but so long as there is sulphur present in the charge, the reduction goes on just as rapidly, so that there is a constant precipitation of copper and liberation of SO_2. It may occasionally happen that the best skimmer of a crew will slightly overblow a charge, and have slag shooting all over the building; but there is usually sharp rivalry between them, and the jeers of his fellows mete out to the offender the best sort of justice. The writer has personally blown and skimmed many charges, and has had just such experiences; and he has learned also that, while many of the men who are competent to handle a charge may never know the reasons why certain things occur, they are sure, nevertheless, of the right method.

The slag should be poured off as quickly as possible and in a steady stream without moving the converter any more than is necessary, otherwise considerable matte may escape. The slag should flow easily, and in a solid dense stream, but if matte is escaping with it, the bottom or back part of the stream will be seen to vibrate, while the slag itself will be less fluid, flowing with the viscosity of molasses. At this point the rabble is shoved into the stream, and if the small crust of slag formed on the rabble is cut away and shows matte, the pouring is stopped and the remainder of the slag is skimmed off. Some slag will still remain behind, but the skimming should in all cases

be as clean as is practicable under the circumstances. It sometimes happens, especially in small converters running on high-grade matte, that the slag will be granulated or only partially liquid. This may be due to a cold converter, in case it is the first charge, or to the lack of heat developed by the charge, due to slow running, which latter may be due to low blast or too large a charge. In such a case the remedy depends upon the cause. If it is due to a cold converter, the addition of from 2000 to 5000 pounds of matte will in most cases cause the slag to become liquid. If the charge is too heavy the blast-pressure must be increased, or a portion of the slag skimmed off in the best way possible, and the finish blow made without skimming clean. The unfused material can be smelted by a small tap of matte after the copper is poured off, the slag poured and another tap taken to furnish metal enough to finish. With very low-grade matte several taps have to be taken to get the required amount of white metal to finish. At Aguas Calientes the converter was eight feet in diameter, the largest copper converter in the world, and on low-grade matte a charge of 8000 pounds for the first tap would be taken and blown to white metal, the slag poured off, and a second tap of 10,000 pounds poured in, blown to slag and skimmed in the same way. A third, and sometimes a fourth, tap would be skimmed before the resulting white metal would be sufficient to stand above the tuyeres until the charge was finished. In this way charges of 45,000 pounds of matte were not uncommon, and the resulting copper would not be more than 40 bars of 200 pounds apiece. With matte as low as 30 per cent. the interior of the converter increased in size so rapidly, owing to the corrosion of the lining, that it was difficult to get a charge to finish. After the vessel had finished the last charge that the lining would stand, a tap of from 10,000 to 15,000 pounds of matte would be put in and blown to slag, or until it was in danger of coming through the side or bottom. This was for the purpose of washing out the copper adhering to the lining after the last charge. The resulting white metal was dumped and used as scrap on subsequent charges.

A considerable quantity of cold matte of high grade is necessary after the skim to keep the temperature of the charge from rising too high. If the charge is too hot the oxide of copper formed is not reduced before it forms a silicate with the lining,

and as this is very infusible, it is chilled by the blast on the tuyeres, resulting in clogging them and making the charge run slow, necessitating much punching. It seems paradoxical, but it is nevertheless true, that above the correct temperature the charge runs much slower, and the tuyeres are hard to keep open. By the addition of white metal in large lumps the difficulty can be avoided.

The use of low-grade matte to cool the charge results in the formation of more slag, owing to the presence of iron, and this slag will, if the quantity is large, result in slow running and shooting out of the vessel, as previously mentioned. If no scrap or white metal is to be had, it is sometimes necessary to throw in a bar of copper to bring the temperature down. The question of temperature must of necessity be told by the appearance of the flame at the nose, and is quite as important as the other indications of slag and finished copper.

In the first part of the blow, or before the slag has been skimmed off, it is not so much a question of temperature as how the converter is taking the blast. If the flame from the nose is the full size of the opening and appears to be leaving the converter with considerable velocity, it indicates that the tuyeres are open and the charge is working rapidly and satisfactorily; but if it goes in puffs, and has a choked and irregular movement, the tuyeres need to be punched until they are free from obstruction. The tuyeres should always be punched before a charge is put in, so that the blast may enter as freely as possible from the beginning. If the charge is sufficiently hot when it enters the converter, it will need very little punching, and will start off with a very dense discharge of smoke and considerable volatilized sulphur. Shortly afterward, probably fifteen minutes, it will have slowed down and will need punching for the next five or ten minutes, when, owing to the increased temperature, it will run freely until time to skim. If the charge is cold when it enters the converter, punching will be necessary from the beginning and for a longer time thereafter than if it were hot. After skimming, the scrap white metal should be thrown in, and any cleanings that may be on hand should be introduced, to the extent of about 10 per cent. of the weight of the charge, if the slag is liquid and hot, but in less quantity if it is viscid; and the blast should be turned on as quickly as possible. Punching will be necessary if the flame indicates

that the converter is not taking air properly. As long as the flame continues to be voluminous, and leaves the nose freely all may be well, but if it assumes a bright brassy color and slows down, the charge is probably becoming too hot, when more scrap should be added according to the requirements, and the tuyeres punched until the proper action is restored. If the flame assumes a light orange color that gradually turns into a darker shade, and then takes on a copper-bronze color, the indications are that it is rapidly approaching a finish, when very close attention is necessary to avoid an overblow and an oxidized charge.

Just the proper point to turn the vessel down is reached, when the little particles of copper ejected from the converter, give the appearance of very fine gauze or lines of a copper color, and when coming in contact with any obstacle they cease to adhere, as they will continue to do so long as they are of matte. The copper adhering to the punch-rod will also indicate very closely the time of finish.

The vessel is then turned down and the granulated slag which will be present on top of the charge, is shoved aside by means of the rabble until the surface of the liquid copper is exposed. If on skimming off a clean surface, the charge shows a bright metallic mirror of copper, the charge is ready to pour into moulds or into a large ladle; but if the surface is covered by a skin of black sulphide of copper, it is necessary to turn on the blast again for a short time.

If considerable slag is present, or if the charge is slightly overblown, the copper will be covered with a layer of slag which will foam and bubble and require some time and considerable cold slag, or cleanings from the floor, to chill around the nose so that the copper can be brought to view. If there is sufficient copper in the converter to stand above the tuyeres, it is possible to completely oxidize a charge, but, although this sometimes does happen, it is through carelessness, or a mistake in judgment on the part of the skimmer. I have known it to happen when copper oxide on a charge already slightly overblown, was mistaken for matte and the charge was kept working long after it should have been poured. Such things will happen, but seldom more than once to the same man; it is not by any means a sign of incompetency, and having once occurred is sure to improve the future service of the skimmer.

If towards the latter portion of the finish-blow, the flame becomes very dark and red, the charge is becoming cold, and the probabilities are that it will not finish without the addition of more matte. If it does finish cold, it will be difficult to pour, and will leave much copper adhering to the lining. In such cases another small tap of matte is put in and blown to slag and skimmed, or at times when the matte is unusually low it is necessary to throw in a large quantity of cold scrap for the purpose of chilling the slag and making it impossible to skim. The reason for this is that with low-grade mattes the corrosion is so rapid and the addition to the copper contents of the charge so small, that instead of increasing the hight of the charge above the tuyeres it is probable that if the slag were poured off it would be decreased. So that it becomes necessary to granulate the slag and force it to mix with the copper, raising the charge above the tuyeres, in order to insure the blast penetrating it until the copper is finished.

The methods that are used to develop heat in a charge that has run cold are; first, the addition of billets of wood; heavy cord-wood is preferable on account of its greater density and the fact that it will, by floating in the charge and by the gas generated, raise the charge level so that the blast will continue to penetrate the metal bath and develop heat, instead of blowing over the top and freezing it. Second, the addition of lump coal, the effect of which is the same as cord-wood. Third, the addition of a small amount of matte, finishing with granulated slag, as described above. The reason for granulating the slag is that in this condition it will not cause the shooting and foaming of the charge described when overblown. This last method is, as before stated, employed in case the matte is of low grade, but if it is high grade, say 55 per cent., the slag may be safely skimmed off, the resulting white metal being enough to compensate for the extra corrosion of the lining, and also to make up the shortage previously existing. If the converter is of the cylinder or Leghorn type, it can be turned back until the tuyeres are brought to a lower level and the blast is forced to penetrate the charge, but with the great majority of converters this is not possible, since turning the converter back beyond a fixed point is prevented by the smoke-stack into which the fumes are discharged. Only the experience and judgment of the skimmer is to be relied on

in such cases, and the remedy must be applied which is best suited to the conditions.

The relative heat-developing power of sulphur and iron is very strikingly illustrated by the action of different grades of matte. A charge of low-grade matte will smelt fully half its weight of granulated slag left in the converter, while a charge of high-grade matte will only add to the difficulty.

The low-grade matte containing more iron has greater heat-developing power, as well as more basic action on the silicious slag, and will bring it to a perfectly fluid condition, while the high-grade matte containing less iron and making a more silicious slag will be unable to smelt the accumulation in the converter. The heat developed in the first portion of the blow, as well as the fact that the iron is all converted to oxide much sooner than the sulphur, shows that the affinity of iron for oxygen is stronger than that of sulphur; and in combining with oxygen iron develops more heat than sulphur. This is also proved by the action of the charge in becoming rapidly hotter in the blow from matte to slag, and gradually colder after the iron has been oxidized and the slag skimmed off. It is a common error often repeated that the heat developed in copper converting is due entirely to the burning of the sulphur, while the fact is that more heat is developed by the burning of the iron.

For this very reason it becomes difficult to convert mattes as high as 65 per cent. Cu on account of the decreased amount of iron and insufficient heat development. It is apparent at once that the presence of the iron is necessary, and that being present, silica must be provided for it to act upon to form a fluid slag.

At Aguas Calientes the lining was made entirely of ore, and this contributed a great deal to the success of converting low-grade matte at that point. If quartz had been used instead of ore the expense would have been too great to admit of 30 per cent. Cu matte being converted. It was very fortunate that an ore of such ideal composition for the purpose was to be had. This ore came from Pachuca, and was mined there in large quantities.

A partial analysis showed: SiO_2 72 per cent., FeO 5 per cent., CaO 0.6 per cent., Al_2O_3 15 per cent. The ore, ground in a Chilian mill with water, was very plastic and did not need the addition of clay, so that it was possible to run with a lining on which there

was a margin of \$20 (Mexican) per ton. Where such ores can be obtained they prove a bonanza to the copper converters.

There are three kinds of finish on converter copper, according to the time the charge is turned down.

The first shows a small amount of regal, and is usually about 95 per cent. Cu, and sometimes expands to such an extent on cooling as to make it exceedingly difficult to get the bars out of the moulds. On this account, as well as the extra time and expense in refining before casting into anodes, it is seldom made, and then only when a charge is too cold to be kept longer in the converter, or on account of weak lining.

The second and most common and desirable of the three is called gas finish, on account of the large quantity of SO_2 which leaves the metal on cooling. This finish shows no regal, but contains SO_2, dissolved in the copper to such an extent that a mould filled with the metal will, after the ebullition has ceased, not be more than one-third to one-half full, and it is necessary to pour into the moulds two or three times in order to make a fair-sized bar.

The SO_2 will remain with the copper as long as it is in a molten condition in the converter, but as soon as the copper strikes the cold mould and begins to solidify, the gas comes off rapidly, and if great care is not taken in pouring into the moulds, the copper will effervesce and run over like soda water.

At times a crust will form over the top of the bar before the gas has escaped from the liquid interior, and then a rupture will take place and a stream of molten copper may be thrown a distance of several feet by the escaping gas. Serious burns frequently and unfortunately occur on this account, and it is rather dangerous to stand near the moulds until the copper is thoroughly solidified. There seems to be a very strong resemblance between the affinity of molten silver for oxygen and this peculiar action of converted copper and SO_2. It does not occur with blister made in reverberatories for the reason that all the oxidation takes place on the surface, while in the converter it goes on much faster and all through the metal bath, some of the SO_2 formed being dissolved in the copper.

The third is called blister finish and exhibits the characteristic blisters on the surface of the bars from which it gets its name. In a charge of ordinary size there is about ten minutes' difference in time between the first and second finish, and about five minutes between gas finish and blister finish. The blister finish contains

a small amount of gas, but usually not enough to cause the copper to decrease much in volume on cooling. It is seldom that a whole charge, especially if it be a large one, will be blister finish. Usually the last few bars are gas finish, while the copper first poured from the top of the charge will be blister. In order to produce blister it is necessary to overblow slightly, and some copper oxide will be floating on the surface as slag.

The distribution of silver in the copper bars, as determined by assaying samples taken from different parts, shows the folly of trying to get a correct sample except after remelting or from the stream as it comes from the converters. The results of two bars sampled at Aguas Calientes are given below:

Blister Finish			Gas Finish		
No.	Part of bar	Assay in troy ounces silver per ton	No.	Part of bar	Assay in troy ounces silver per ton
1	End	358.1	1	End	231.2
2	End	403.4	2	End	247.4
3	Top	608.	3	Top	451.7
4	Bottom	366.2	4	Bottom	203.6
5	Side	393.9	5	Side	252.9
6	Side	372.5	6	Side	235.9
7	Fin	423.6	7	Fin	351.4

In order to get a sample of a carload lot of converter copper, it is necessary to take a sample of each charge, and mark the number of bars on the ticket that is to go with it. To weigh each charge separately will be the correct method; but as the bars will average about the same, the number of bars can be used instead. A number of these charges are bunched together to make a carload, and the samples are cut up into small pieces and as many grammes taken from each as there were bars in the charge. These weighed portions are put together in a large clay or graphite crucible, melted in a blacksmith's forge, and granulated by pouring slowly on to a board set obliquely in a bucket of water. The stream of copper should strike the board about six inches above the water, and should fall a distance of about two feet before striking. It will glance off in fine shots and, chilled by

the water, will be found as bright granules, when the water is poured off.

Later Experience

In the previous editions of this book will be found the statement that it was impossible to convert copper matte in a basic-lined converter. This statement was made on insufficient evidence and I am obliged to say that it has been demonstrated that a converter lined with magnesite brick and supplied with the necessary amount of silicious material is more economical for

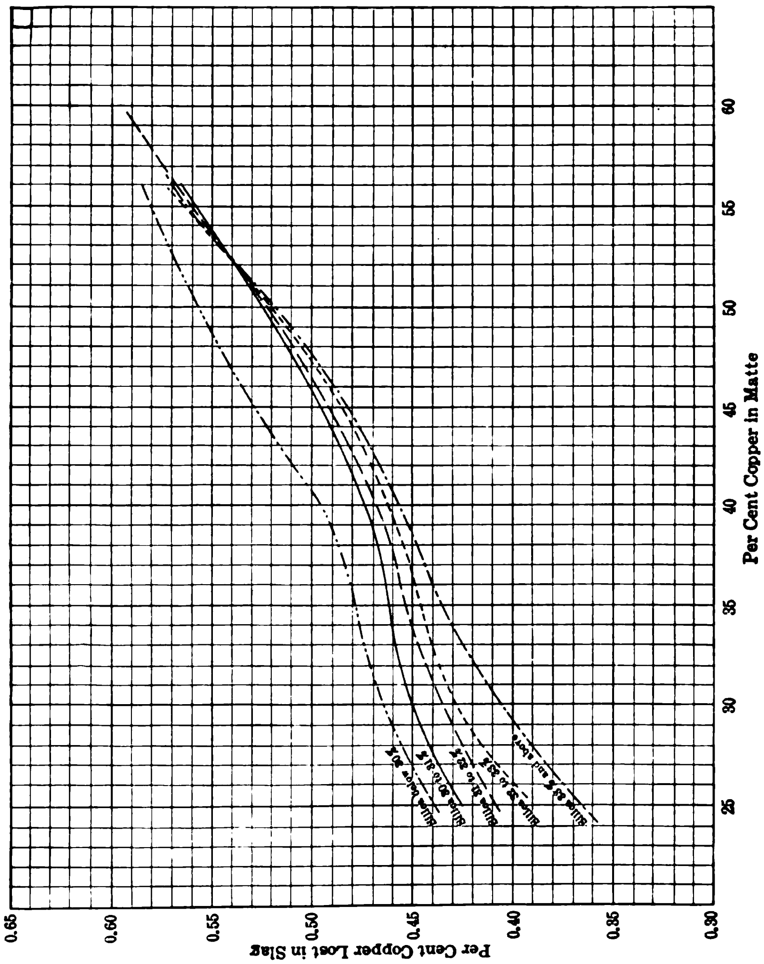


FIG. 30. — Effect of Silica on Copper Contents of Slag. W. A. Heywood.

converting low-grade matte than the converter with silicious lining. There are several reasons why this should be so. The low-grade matte means that the slag losses in the smelting-furnace will be much less than if higher-grade matte has to be produced to meet the requirements of silicious-lined converters. This is a most important item, as a difference of $\frac{1}{10}$ per cent. of copper means 2 pounds per ton of slag, and as this copper would be produced for the same expense as the lesser quantity if it were lost, it is right to say that it is worth the full market value less freight and refining charges. This might mean anywhere from 25 to 50 cents per ton of slag, depending on the price of copper. In case a plant produces several hundred tons of slag per day this would mean that a terrible sacrifice of copper is made, which could be saved by making low-grade matte, say 20 per cent., and using magnesite-lined converters in place of silicious-lined converters. Another reason is that no roasting or second smelting operation would be necessary in order to increase the grade of the matte. With proper appliances the large amount of converter slag can be poured back into the smelting furnace, or into the settler, and cleaned of its copper. A silicious ore that would not do for converter-lining can be fed into the converter to supply the silica, and can serve the double purpose of smelting the ore and acting as a flux for the iron in the matte.

A blast-furnace running on a charge producing a large amount of low-grade matte will run faster and with less trouble, and treat more ore with less fuel, than if obliged to produce a higher grade of matte. A plant with magnesite-lined converters would not need so many shells, and the traveling crane may be dispensed with, greatly reducing the first cost of the plant. Considering all its advantages, which have been pointed out by Mr. Heywood, it is remarkable that there has been no general application of the process. The cause of the failure of the earlier experiments with magnesite-lined converters was the fact that 55 to 60 per cent. copper matte was used in these experiments. Mr. Heywood found that 30 per cent. matte did not work so well as a lower grade. This is fortunate, as the lower-grade mattes make cleaner blast-furnace slags. I have been informed by Mr. Rohn that they have abandoned the use of the magnesite-lined converter because they cannot make low-grade matte and the magnesite lining will not work on high-grade matte.

XIV

DESIGN OF CONVERTER PLANTS

When Copper converting was introduced in this country the converters were made stationary, and in order to reline them they were filled with water, as in the old plant at Anaconda. But the experience gained showed that this practice would have to be abandoned. As it was impracticable to wait for them to cool in the stands, it became necessary to remove them and place a fresh vessel in position, so that work could go forward without serious interruption.

The Parrot plant was constructed with converters 60 inches diameter by 8 feet 6 inches high, which were removed on a car. The lifting device was four jack-screws, one at each corner of the car, and the heads of these screws impinged on lugs riveted to the converter. The converter was turned on its back by the hydraulic cylinder, the car was run under the converter, and the jack-screws applied until the weight of the converter was lifted off the trunnions. The trunnions were then uncoupled at the flange joint, one on each side between the vessel and the stand, the tips of the bearings remaining in the stand. The car bearing the converter was then run out to the relining house, and another car with a converter lined and dried was run in place, and after coupling the trunnion tips, was ready to receive a charge. This method was discussed for the Anaconda plant, but it was fortunately decided that a traveling crane would be better, and the plant was built according to the plans reproduced in the appendix.

One of the difficulties encountered in this plan was to get the converter and cupolas to deliver their smoke into the same flue. It had previously been the practice to have the converters turn down towards the cupola to receive the charge, and turn up and blow away from it. This would have been impossible with a traveling crane, and the difficulty was overcome by putting two converters to each cupola and making them turn down,

away from the cupola, to receive the charge and turn up and blow toward it into the same flue.

The first section of the spout from the cupola well was straight, and emptied into a broad section which was movable and which divided into two spouts, one running to the right- and another to the left-hand converter. These spouts were mounted on wheels and could be swung around to either converter when a charge was needed. By means of a few shovels of clay the matte could be diverted to either spout, which would carry it to the converter. The trunnion coupling which was selected for use after many designs had been made is worth special consideration, since it is very strong in construction and can be coupled up or uncoupled in about 30 seconds. The trunnion ring remains on the converter and is taken out with it. The tips of the trunnion ring are conical in shape, with the large ends farthest from the vessel. These cones fit into pockets in the trunnion tips, and are fastened to them by heavy gibs and keys, as well as bolts on the bottom (see Plate V, appendix).

The dimensions of the vessels adopted were 72 inches diameter by 10 feet high. It would have been better had they been made 7 feet × 14 feet high, but at that time the large converters at Great Falls were not giving satisfaction, although later they improved wonderfully under different management. The discouraging reports circulated had the effect of placing a limit on the size of the vessels at Anaconda.

Since that time it has been the experience of the writer that very large converters do not do as good work on high-grade matte as smaller ones, while on low-grade matte they do much better. The extremes of size thus far in use are: Parrot, 58 inches diameter by 8 feet 6 inches high; Aguas Calientes, 96 inches diameter by 16 feet high. The converters at Aguas Calientes are illustrated in Figs. 32 and 33.

The main objection to large converters on high-grade matte is that the slag is very frequently granulated and cannot be poured off; while with the smaller converters, as at Anaconda, no such difficulty was ever experienced; in fact, the trouble was to get the matte high enough in grade so that the linings would last six or seven charges.

As the size of the vessel increases, the thickness of the lining also increases rapidly, and since it is very porous, the loss of blast

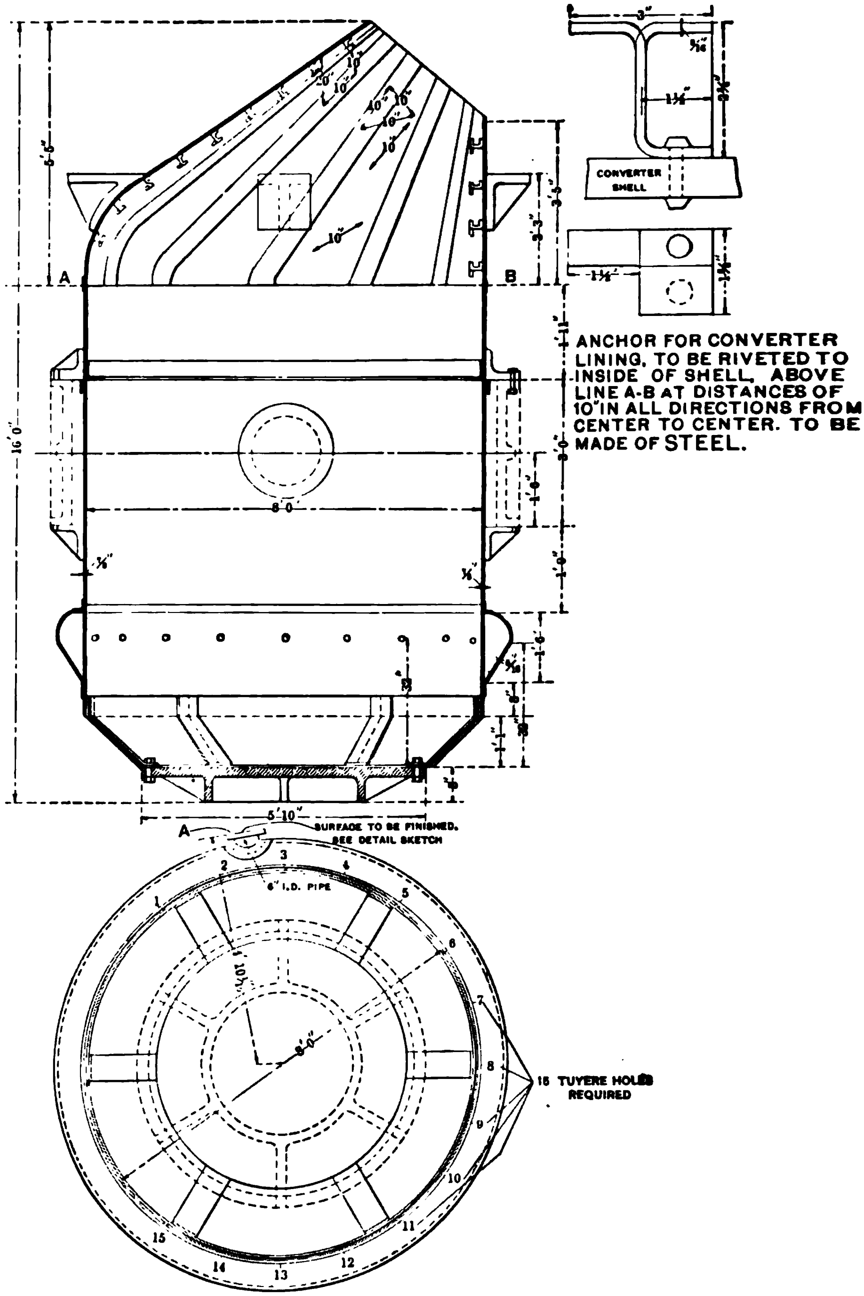


Fig. 31.—Copper Converters at Aguas Calientes.

through the lining also increases. A point would soon be reached where with increased thickness of lining the decreased efficiency of the blast would more than counterbalance all the benefits of large vessels. The loss of blast through the lining depends a great deal on how the latter is put in. If it is tamped in around a cone the loss will be less than if it is simply built up by hand. But tamping linings around a cone is much slower and more expensive, and tests made on the life of such linings did not show that they lasted any longer than those put in by simply pounding the wet material into place with the hands.

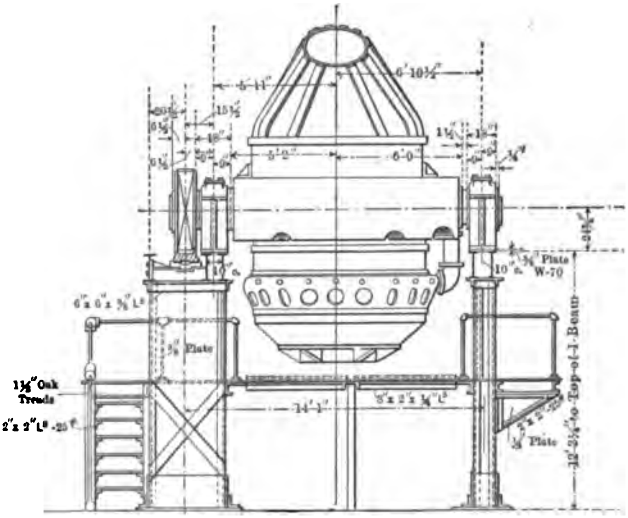


FIG. 32.—General Arrangement of 8-foot Diameter Copper Converter.

There are two methods of arranging a converter plant, either of which is good, and the choice between them depends upon local conditions. If the tonnage is large the crane plant is by far the better, and the size of the vessels should not be less than 7 feet in diameter by 13 feet high. If the tonnage is small the plant should be constructed on about the same lines as the Aguas Calientes plant, but with this change in the converter: that the joint in the lining should be at the top of the trunnion ring instead of at the bottom. Of the crane plant nothing in addition to what has been said of the Anaconda plant is necessary, but of the handling by cars at Aguas Calientes this can be

said: that the plant will cost much less to construct and a much larger converter can be used than could be handled by any crane suitable to a small converter plant. Vessels of this large size when freshly lined, weigh fully 40 tons, and the buildings and crane necessary to handle such a great weight would cost $30,000 more than the other plan.

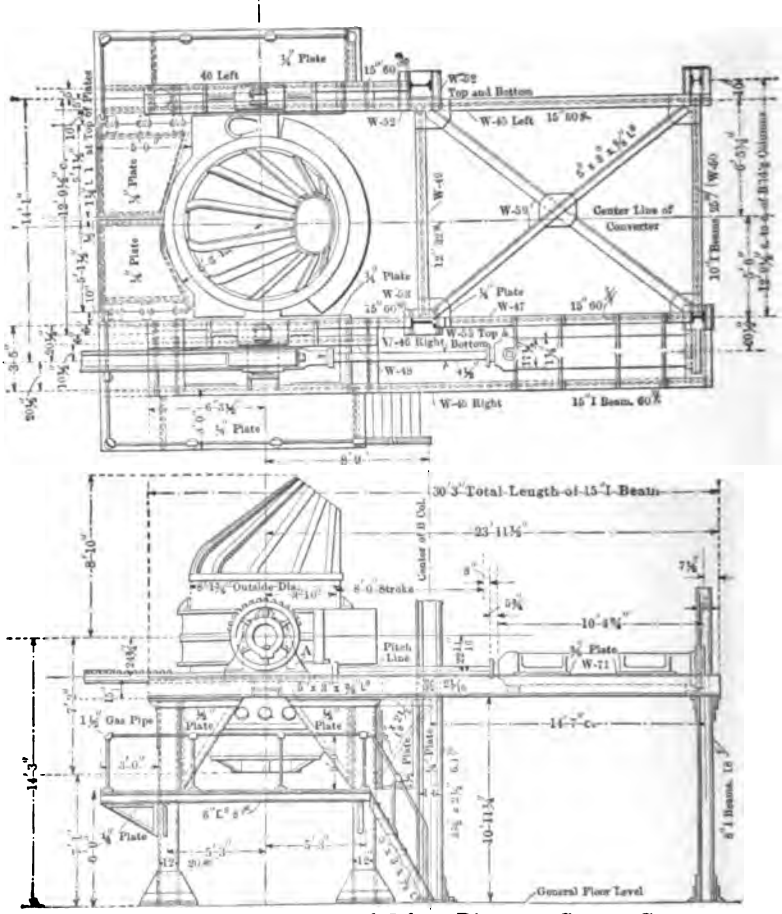


FIG. 33.—General Arrangement of 8-foot Diameter Copper Converter.

The practice at the Aguas Calientes plant was similar to that of most steel plants, where the vessels and bottoms are removed on a car, the car with the vessel on it being lifted into position

by a hydraulic piston about 15 inches in diameter, acted upon by water pressure of about 500 to 600 pounds to the square inch. The top end of the piston is fitted into a very strong frame, which carries a section of the track long enough to allow the wheel-base of the cars to be moved forward or backward a few inches. The converter is put into place in two sections. The

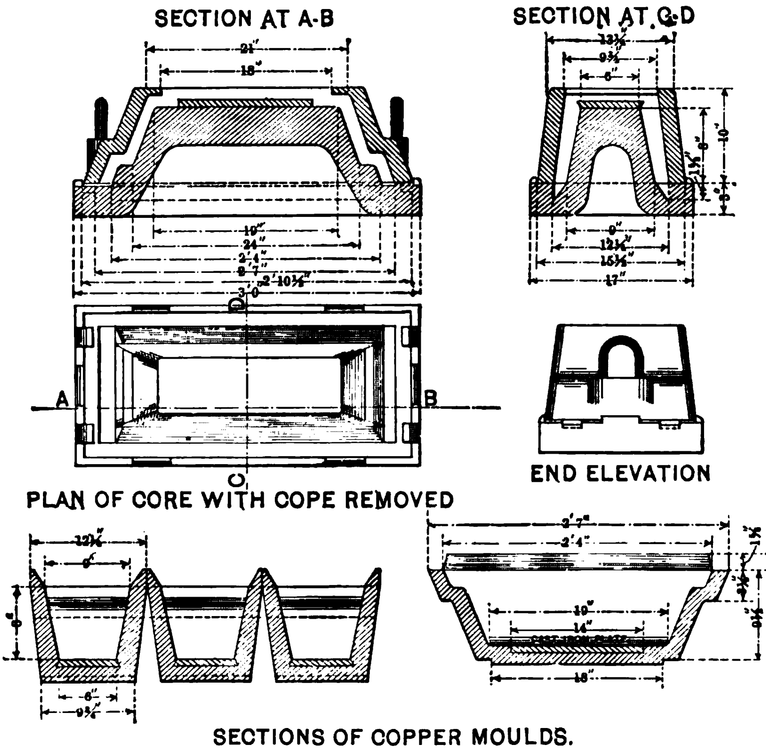


FIG. 34.—Cope and Core Apparatus for Making Copper Moulds.

top section, being cleaned of slag about the nose, is then taken by the car onto the elevator below the trunnion ring, hoisted into place and bolted to the ring in an inverted position. It is then turned upright by the hydraulic cylinder attached to the trunnion ring and the bottom section, fully lined and dried, is run under, hoisted into position and bolted to the trunnion ring as well as to the top section. The remainder of the lining must then be put into the nose, and as this is a very long,

tedious job, the less of it there is to do the sooner it will be done. The mistake was made of cutting the converter in two below the trunnion ring instead of above, or at the upper edge. The bottom section should be the largest piece of the shell, so that as much lining as possible may be put in before the top is put on. The idea in constructing the plant in this way was that the bottoms were the only portion of the lining that were corroded, and that they could be removed and renewed just as is done in steel practice. This idea, of course, is wrong. The lining is corroded wherever the slag and matte touch it, except right at the nose, where the slag thrown out freezes, and causes the opening to grow smaller. Owing to the unfortunate circumstance that a lining 2½ feet thick, of clay and quartz, with nothing to support it, will fall out, the top section could not be lined bottom side up and then turned upright to receive the bottom without losing all the lining in the upper section and causing a wreck that would take several hours to repair. If, as stated, the joint had been made at the upper side of the trunnion ring instead of at the lower, the lower section would have been increased by that amount and the quantity of lining necessary to be put in after the cap had been put on greatly reduced. Owing to the length of time necessary to pass 15,000 pounds of lining in at the nose, and to put it into place, as well as the time required to dry the lining and heat the vessel to a point where it would be in a fit condition to receive a charge, the time in which a change of vessels could be made was unusually long. From ten to fifteen hours were required as against five minutes at Anaconda, from the time a charge was poured until another could be run in. If the joint had been in the proper place a change could have been made in about four hours, including firing the fresh lining.

In other respects this converter plant was a good one, and the large vessel was certainly a great improvement over a smaller one for the treatment of low-grade mattes.

Moulds for Converter Copper

The use of cast-iron or steel moulds for converted copper is rather expensive, and it has been the practice at many places to make the moulds of copper. This is done by pouring refined

copper into a mould made of two ells clamped together, and then plunging a core-bar into the metal bath, which would make the mould. It involves considerable expense for labor and supplies, and the experiment was tried of making the moulds direct from the converter. The moulds could be made very easily by means of the cope and core apparatus shown in Fig. 34, but when in use the stream of copper from the converter would strike on one part of the bottom until it became hot enough to weld, and the result would be that the bar and mould could not be separated on account of a union covering a space of about 1½ inches in diameter. If the copper being poured was very hot it would bore holes into the copper moulds as quickly as would hot water poured upon ice, and even after the difficulty with the bottoms of the moulds had been remedied by means of the cast-iron plate placed on top of the core before casting, great care had to be taken that the stream did not impinge on the sides of the moulds.

The cope and core were placed on top of an ordinary slag-pot, preferably one of which the bowl was cracked, so that it could remain there without being changed. The cast plate shown was then placed on top of the core, and the pot was run under the converter nose. The copper was poured in slowly to allow the SO_2 gas to escape, and the space between the cope and core to fill up solidly. The pot was then pulled to one side, the cap lifted off by means of the traveling crane, and a bar with chisel point was driven between the copper mould and the cast core at one end. A couple of men would then pry down on the bar until the mould came away from the core, or until the mould and core were raised from one side of the pot, when a few sharp blows from a sledge on the raised end of the core would bring them apart. The cast-iron plate came away with the copper mould, and served the double purpose of protecting the core while making the moulds, and of preventing the welding of bars to the moulds while in use.

This scheme worked very satisfactorily, and copper moulds with cast-iron bottoms were used entirely. It was necessary to make from 12 to 20 each day owing to their breaking after about five to seven days' use. The scrap or broken ones were, of course, available as bars for the casting furnaces.

The life of these moulds depended a great deal on the kind of finish copper used in making them; moulds made from blister

finish being found much better than these from gas finish. While making moulds two or three cores on pots were used at one time, one filling while the others were stripping or were being washed in limewater to prevent them being burned. By the use of a few extra men and a traveling crane twenty moulds could be made in an hour.

Later Experience

In view of the success of the magnesite-lined converter on low-grade matte, and the general advantages of the process, as outlined in a previous chapter, it is hardly necessary to add that the chief interest attached to the foregoing description of the designing of a converter-plant, is historical. The best way to treat ores is that way which yields the greatest profit, and therefore the magnesite-lined converter should be used on low-grade copper matte.

In case the ores contain nickel the silica lining seems to be necessary, as was pointed out in a previous chapter. The failure of the magnesite lining on copper-nickel mattes, is attributed to the reaction between nickel oxide and nickel sulphide being harder to bring about than the similar reaction between copper oxide and copper sulphide. It required a high temperature in the absence of the silicious lining, and at the higher temperature the magnesite lining failed.

XV

LINING A CONVERTER

IT IS unfortunate that in most places quartz and clay are so entirely distinct. If, as at Aguas Calientes, they are both contained in one ore, the lining is much more homogeneous and compact. The linings in Montana are made from quartz which has no plastic or adhesive qualities and is very refractory, and a fat, sticky clay, which is not refractory, and which melts away from between the particles of quartz, allowing them to drop off and mix mechanically with the slag or copper. This probably accounts for much of the difference in the composition of the converter slag at Anaconda and Aguas Calientes, which averaged as follows:

	SiO_2	FeO	Cu
Aguas Calientes	25 per cent.	62 per cent.	3 per cent.
Anaconda	36 per cent.	49 per cent.	3 per cent.

The matte at Anaconda would average 55 per cent. and at Aguas Calientes not more than 35, and part of the difference in slag composition is due to the fact that high-grade matte makes more silicious slags than low-grade.

The introduction of silica through the tuyeres was not attempted at Anaconda, for the reason that it was apparent to the writer that so much silica as would be required, could not be blown in within the short time allowed, and, second, because it would act as a sand-blast and ruinously cut into the ironwork of the converter in a very short time.

Even if the silica could be introduced in this way it would still produce another trouble in the converter. The small particles of silica, being cold, would combine very imperfectly with

the iron oxide and produce granulated and pasty slags, which could not be skimmed or poured out of the vessel.

About 60 tons of quartz and 7 tons of clay were consumed daily for lining, and two sets of crushers and rolls and three Chilian mills were kept going day and night to grind the material for lining. The mixture used in charging the Chilian mills was 40 shovels of quartz to five of clay. This was ground together with hot water in winter time, and cold when the weather would permit, until the whole mass was reduced to the consistency of a stiff mud and the quartz pulverized to the size of peas. In this condition it was discharged from the mills into a stock-pile, or direct into wheelbarrows, to be taken to the converters in process of lining. Before passing to the liner it was pounded with the shovel until it adhered thoroughly together, and was then cut into cubes by the shovel and passed into the converter. As soon as the converters were removed from the stands, the nose section was taken off by removing the keys from the key-bolts and lifting with the crane, when the joint would generally break, but if it did not, a wedge driven in would start a fracture. To hasten the cooling the interior would be sprinkled with water from a hose. The skin of slag was then cut out and the bottom section relined, after which the top was put on by the crane and the lining finished as high above the joint as possible. The converter was then taken by the crane to a place where there was a blast connection through a 3-inch hose to a branch from the main supplying the blast-furnace. A fire was kindled with oily waste and dry wood, and shortly afterwards coke was put on and the blast turned in. The fire was kept up until the converter was required for use in the stands. From three to four hours were needed to dry and bake the lining sufficiently to insure it from falling out when the vessel was inverted. It was found that if the fire was allowed to go out the lining would contract so much that the probabilities were it would collapse when put in the stand and turned upside down.

The life of a lining should not be measured so much by the time or charges as by the copper produced and the matte converted. The production of a lining is dependent on the composition of the clay and quartz, as well as upon the grade of matte converted. If the clay is not plastic or the mixture is poorly ground, the lining may collapse after a single charge. On the other hand,

if all things are working to the best advantage and the grade of the matte will average 55 per cent. Cu, the first charge for a 6-foot vessel should produce about eight to twelve bars of copper of about 250 pounds to the bar; the second charge from ten to sixteen bars, the third from sixteen to twenty, and the fourth from eighteen to twenty-eight, the fifth from twenty to thirty, the sixth from twenty to thirty-five, and so on until the lining becomes too thin to stand another charge. A production of 100 bars is a good run for a lining on 55 per cent. matte, and all statements to the contrary should be taken with considerable doubt. In one work on the subject a statement is made that the linings are usually exhausted after the ninth charge. It is safe to say that with the size of vessels in use at the Parrot works, where this remarkable work was done, all the linings that have made nine charges in the past two years can be counted on the fingers of one hand. At Aguas Calientes, where the vessel was 8 feet in diameter and the lining 2½ feet thick at the tuyeres, one charge of 30 per cent. matte weighing 40,000 pounds would, if added 8000 to 15,000 pounds at a time, finish about forty to forty-five bars and corrode the lining so much that a second charge could not be finished. With such low-grade matte the second charge would only be blown to white metal and skimmed, the 80 per cent. copper matte being poured into beds and returned to the next charge as scrap. A single lining would, including the wash-out, convert about 50 tons of 50 per cent. matte.

The experiment was tried at Anaconda of turning the converter on its back in the stand and putting in a large patch of green lining wherever needed, and allowing it to dry thoroughly before a charge was run in. By repeating this operation several times twelve charges were finished with a total production of 212 bars for a converter 60 × 60 inches square. Aside from the large vessels at Aguas Calientes this is the largest production for a single lining that I know of. The largest charges thus far finished were probably made at Great Falls, where according to report something over 80 bars, or about 16,000 pounds, of copper have been poured. The largest single charge at Aguas Calientes was 75 bars.

Later Experience

The present method of lining converters with silicious linings is by tamping the material around a form in the lower section of the converter by means of a machine tamper operated by compressed air.

The lining is in all cases made of some silicious ore, when such material is obtainable, and at many places where the ores have to pass through a concentrator before going to the smelter, the slime from the ore is used to bond the converter lining in place of clay. A magnesite lining can be put into any converter if magnesite brick and magnesite powder and magnesium sulphate are obtainable. These materials can be purchased from the Harbison Walker or the Federal Refractories Co.; the only two sources of such supply of which the writer knows.

It is not advisable to duplicate the converter-shell of heavy steel rings which was used in the first successful plant in Butte, as it is certain that only the magnesite lining proper has any bearing on the success.

To put a magnesite lining into an ordinary converter-shell, it is advisable to remove nearly all of the old lining and tamp in a bottom to such a depth that when a magnesite brick is laid on edge it will still be about eight inches below the tuyeres. A mortar of magnesite powder wetted with a saturated solution of magnesium sulphate, is used to lay the brick in. The magnesium sulphate sets well and makes a strong bond. The side walls are built up 9 inches thick on the bottom, and silicious lining is tamped between the magnesite brick and the converter-shell. The entire interior is lined in this manner, and then when the converter is heated to a high temperature, it is ready for a charge.

In blowing a charge on this lining it is necessary to add enough silicious material to make a slag of 30 per cent. SiO_2. If the silicious material is not added fast enough the magnesite lining will be destroyed. The addition of the proper amount of silicious material is, therefore, essential to the success of the lining. Regarding the behavior of the converter at Butte the following extract is made from a letter from Mr. Rohn, manager of the Pittsburg & Montana Co.:

"The magnesite brick used by us in converter lining were made by the Harbison Walker Refractories Co. The linings cut

away most rapidly at the tuyeres, and required most frequent repairs at this point. We used two thicknesses of brick, with practically no filling behind them. The steel was eleven inches thick.

"Early in the experiments an attempt was made to run the converter with the metallic shell exposed, and water was sprayed upon it to reduce the temperature. The result was a complete destruction of a part of the shell, necessitating replacement of the damaged section. Thereafter the converter was always shut down immediately upon discovery of a break in the lining. We feel confident that the thickness of the metallic shell is in no way responsible for the difference in behavior of your converter and ours. It is barely possible that the filling of non-conducting material in the form of ordinary converter lining between the magnesite brick and the metal shell may have some influence, though we think it more probable that the difference is due to the amount of silica in the material treated. It was found in our experiments that by feeding a sufficient amount of silicious ore into the converter, a condition could be maintained whereby the brick lining was slightly coated and, therefore, to some extent protected; while when treating a very basic matte and cutting down the silica, the brick were found to run very clean and to cut away more rapidly. Our linings lasted six weeks, sometimes longer, without repair. Our experience showed us that the wear on the lining was greatest when first starting a new one, and at such times we were particularly careful about charging sufficient silica.

"If we should decide to experiment with the magnesite brick next to the metal lining, without the use of ordinary converter lining, account must be taken of the expansion of the brick. We found this to be sufficient to burst the heavy stay-bolts in the 11-inch shell."

XVI

CASTING ANODES DIRECT FROM CONVERTER

THE CASTING of anodes from the converter had been attempted at the old plant at Anaconda, and was successful enough to indicate that it could be done with the assistance of a traveling crane. When the new plant was well started and the men had become accustomed to the use of new appliances, the casting of anodes was carried on for some time, although eventually abandoned because of an objection made that the impurities in the converter anodes caused the electrolyte of the refinery to become too impure, it being stated that the cathode copper from such anodes would be low in conductivity and unfit for wire bars. However, before this objection came from the refinery, it was demonstrated that anodes of reasonably uniform weight and density could be cast at a saving of about 0.35 cents per pound, or $7 to the ton. The anodes were cast on edge between cast-iron moulds grouped together so that the face of one mould would be the back of the next, the set being held together with iron clamps with springs to allow for expansion when filled with copper. Owing to the rapid chilling of the copper when it was poured into the moulds, it was necessary to be able to pour along the entire length of the mouth of the mould, as well as to change quickly from one mould to the other and back again. The escaping SO_2 gas would cause the copper to shrink in the moulds, and the anodes would be hollow unless filled a second time. To overcome these difficulties a car with a movable table-top on differential rollers was designed, and the set of moulds put on this table. All the peculiar difficulties were overcome except the warping of the cast-iron moulds by the heat from the copper. This caused the anodes to be somewhat irregular in thickness and weight, owing to the fact that the moulds, after being used for a few days, would buckle so much that they could no longer be drawn close together by the clamp. This variation would amount to 20 per cent. of the weight of the anodes, in

moulds made to cast plates of equal thickness. The anodes were taken from the moulds to a large Gate shear, where the ragged upper edge was sheared off, and two holes punched in the ears by which they were suspended in the tanks.

It was found that plates could not be cast less than 1 inch thick on account of the chilling of the copper before the mould had been filled in all parts. Anodes 1⅛ inches in thickness of the dimensions required would weigh 230 pounds made of converter copper, and if made of cast copper would weigh considerably over 300 pounds, the difference in weight being due to the porous character of the converter copper, caused by the escaping SO_2 gas. Anodes of unpoled and converter copper, which contain more impurities than were contained in these, are used at other refineries, but with what success as regards the conductivity of the cathode copper, I am unable to state.

The average assay of samples of converter copper showed 99 per cent. Cu, with silver varying from 80 to 120 ounces and gold from two-tenths to five-tenths ounce to the ton. There was considerable SO_2 gas retained in the pores of the anodes, which it was stated, was converted into sulphuric acid in the electrolyte, resulting in the increase of acidity of the solutions and rendering unnecessary the addition of free acid. This, as well as the porous character of the converter anodes, which later would present greater surface to the action of solutions, should be considered as points in their favor.

If the anodes are cast from the converter in open moulds, as at Great Falls, the thickness and weight are subject to greater variation than if cast on edge. The stream of copper, as it comes from the converter, has to be broken up, and deflected to a different part of the mould by allowing it to fall on a board held by an attendant. If this is not done the copper will set before the lugs have been made, and the anode will be much thicker in the middle than on the edges.

There is a much greater production of scrap in the refinery from converter anodes than from cast anodes, owing to the imperfections of the former.

Later Experience

At several of the large copper smelters the copper is poured from the converters into a ladle and from the ladle into a rever-

beratory furnace, where it is subjected to a refining process before casting into anode moulds.

This saves time on the converters and increases their capacity, besides saving considerable time and expense in reheating the copper. The copper is run from the casting furnace to the moulds of a casting machine. As the level of the copper falls in the furnace a part of the baked clay breast is chipped away, allowing the stream to issue at a lower level.

The stream of copper pours into a small ladle in which an iron ring is allowed to float on the melted mass. All the chips and dirt stay inside the ring, and if necessary the ladle can be changed. The stream overflows from the ladle to the moulds as they pass on a track.

XVII

COST OF PRODUCING COPPER AT ANACONDA IN 1895

IN EXPLANATION of the following figures, in case they should not seem clear and intelligible, it is only necessary to state that the losses in percentage are on a basis of the material charged to the different departments. To get the copper marketed it is necessary to deduct them in their order from 100 per cent. in the ore after multiplying by the percentage delivered to that department.

For example, if 18 per cent. is lost in dressing and 9 per cent. in smelting, then 100 — 18 = 82 per cent. delivered to smelter; 82 × 9 per cent. = 7.38 per cent. of Cu in the ore lost in smelting; 82 per cent. — 7.4 = 74.6 delivered to converter; 74.6 × 3 = 2.238 per cent. of Cu in ore lost in converting; 74.6 — 2.2 = 72.4 delivered to casting; 72.4 × 1 = 0.72, and 72.4 — .7 = 71.7 delivered to refinery, etc.

In the same way the costs are figured, only the cost for the department is divided by the per cent. of Cu marketed. Starting at 100 per cent. and working backwards, 1 per cent. loss in melting would have the cost divided by 99 per cent., 1 per cent. loss in casting by 98, etc.

	Per cent.
Copper in ore	100
Loss in dressing	18
Delivered to smelter	82
Loss in smelting	9
82×0.09 = 7.38 (say 7.4); 82 − 7.4 = 74.6	
Copper delivered to converter	74.6
Loss in converting	3.0
74.6×0.03 = 2.238 (say 2.2); 74.6 − 2.2 = 72.4	
Copper delivered to casting department	72.4
Loss in casting	1.0
72.4×0.01 = 0.724 (say 0.7); 72.4 − 0.7 = 71.7	
Copper delivered to refinery	71.7
Loss in refining	0.5
72.4×0.005 = 0.3585 (say 0.3); 71.7 − 0.3 = 71.4	

Copper delivered to melting department	71.4
Loss in melting	1.0

$71.4 \times 0.01 = 0.714$ (say 0.7): $71\ 4 - 0.7 = 70.7$

Copper finally recovered from ore	70.7

Cost of dressing 0.53 per pound of copper in concentrates.

Making the calculation in the same manner on the basis or the copper contents of the concentrates delivered from the dressing works, it appears that 86.2 per cent, is recovered, i.e., the loss in smelting and converting is 13.8 per cent. The cost of dressing per pound of copper marketed is consequently: $0.53 \div 0.862 = 0.614$. In a similar manner the cost of smelting per pound of copper marketed works out: $2.035 \div 0.945 = 2.153$; cost of converting matte to blister: $0.6870 \div 97.5 = 0.705$; cost of casting: $0.35 \div 98.5 = 0.356$; cost of refining, 1.00 cent; cost of melting, 0.40 cent, cost of mining: $2.2 \div 70.7 = 3.112$. The recapitulation is as follows:

Cost mining per pound Cu sold	3.112 cts.
Cost concentrating per pound Cu sold	0.614
Cost smelting per pound Cu sold	2.153
Cost converting per pound Cu sold	0.705
	6.584
Cost casting per pound Cu sold	0.356
Cost refining per pound Cu sold	1.000
Cost melting per pound Cu sold	0.400
Total cost per pound Cu sold	8.340

To each pound of copper there is recovered an average value of about 4 cents in precious metals.

The building at Anaconda was designed for twelve converters running and thirty-six shells, three shells to each stand. There were to be six cupolas, one to each two stands, and two blast-furnaces to work over the converter slag. Three No. 7 Roots blowers were required to furnish the blast for the cupolas, for the blast-furnaces, and for drying out the converters.

There were four blowing engines with a capacity of 2500 cubic feet each per minute, and two with a capacity of 8000 cubic feet each, making a total capacity of 26,000 cubic feet per minute, or about 2200 cubic feet per minute for each converter in operation.

To furnish the steam there were eight fire-box boilers with shells 72 inches by 18 feet besides the fire-box. Each had a steaming

capacity of 160 indicated horse-power, making 1280 horse-power in order to turn out 11,000,000 pounds of copper per month. This would be the maximum if all were running at one time, but out of twelve converters seldom more than eight were running, and as the speed of the engines was regulated automatically by the air pressure, the maximum power was seldom required. The blowing engines were all furnished with Corliss steam valves, and the four small ones with Corliss air valves. The large engines were built according to specifications with gridiron slide-valves for the air cylinders, and it was found that these were much better than the Corliss valves. The cost of this plant was about $400,000.

XVIII

VICTORIA MINES, ONT.

Later Experience

Practice of Copper Nickel Smelting at the plant of The Mond Nickel Co. — The crushed ore from the mine is hand-sorted on a bumping screen-table. The crusher is a 9 × 15 Blakes, and discharges onto the bumping screen, which is 19 feet long by $3\frac{1}{2}$ feet wide, made of $\frac{5}{16}$-inch steel plate with $\frac{3}{4}$-inch holes drilled 1 inch center to center. The table has a drop of one in ten, and is hung by parallel rods from the joist, and is actuated by a cam. The fines that pass these holes drop into pockets beneath the screen while the coarse passes over the end into separate bins. The rock pickers drop the rock into pockets intermediate between the fines and coarse, and the various products are loaded into separate buckets and sent over the Blichart Aerial tram to the roast yard, or to the rock dump under the tram line.

The places near the rock houses where the tram-line passes over deep ravines, are used for dumping rock. The dump is started by placing a temporary platform alongside of a tram tower and stationing a man there to dump the buckets as they pass. When the pile has become high enough, the man uses it to stand on instead of the platform, and continues to use it until the next tower is reached. An automatic dumper could be used, but the man would be needed to change it every time ore was to be shipped through to the roast yard, and it might be the cause of wrecks and dumping ore on the rock pile.

At the roast yard the fines and coarse have to be dumped at different places and the buckets have to be stopped while dumping, so that an automatic dumper is of no use. A temporary platform upon legs made of poles is built at such a hight that the dumper walks alongside of the bucket between the ropes, so that he can detach the bucket from the traveling rope without allow-

ing the traveling rope to slip out of the clutch. While the bucket is being dumped, the traveling rope slides through the clutch, without pulling the bucket along the line.

The towers are about 25 feet high and the buckets hang at about 18 feet from the bottom of the roast beds, which are built at right angles to the tramway. A track is mounted on a temporary trestle made of poles, at such a hight that the tram buckets can dump into a car on this track. The car runs on the track at right angles to the tram over the top of the bed and carries the ore from the tram to the end of the bed where it is dumped. The bed is made about 40 feet wide at the bottom and 120 feet long. The hight is variable as the tram ropes sag down in the middle of the spans. Ordinarily the beds contain, when finished, 8 to 10 feet of depth of ore, on 2 to 3 feet of wood. The average number of tons in a bed is 2500. The amount of wood required depends upon the moisture in the ore, the quality of the wood, the general weather conditions, and the amount of rock associated with the ore. About 100 cords is required to give a good roast to a bed of the size mentioned. This would mean an expense for wood of $250 for 2500 tons of ore or 10 cents per ton.

The coarse ore is dumped in the center, and the fines on the outside and top of the piles. The wood is ignited all around the margin by a liberal use of kindling and kerosene, and is kept burning as rapidly as possible until the fines on the side run down and smother it. The air is shut off after about six hours' burning, when slow combustion begins. The wood must be well fired by a good start or the ore will fail to ignite and the entire bed or a large portion of it, remain in an unroasted condition. On the successful roasting of the ore depends all the subsequent operations, and the cost of smelting and converting can easily be doubled by failure to get a good roast. It is, therefore, advisable to make all the conditions as favorable as possible for roasting by using enough wood, by keeping rocky ore out of the beds, and by all other precautions. Well roasted ore can carry as much green ore on the furnace charge and still make a good grade of matte, while poorly roasted ore will produce more low-grade matte than can be converted and will not admit of the use of any green ore on the furnace charge while it is being smelted.

After burning for sixty to ninety days these piles are ready

for smelting. The ore may be hot and may require spraying with water if the smelting is running too close upon the roasting. This is bad practice but is frequently necessary when a larger quantity is required to be smelted than the stock of ore on the roast yard will justify. To smelt 150 tons per day, about 20,000 tons should be in process of roasting. The beds need attention during burning. If the crust of fines becomes too hard it may require loosening up, and if chimneys form, they must be closed to prevent matting of a large quantity of ore.

A track is laid down at each end of the beds, on which two-ton skips are hauled by horses. The skip is run into the end of the bed with the open end nearest the ore. The entire bed is reloaded into these skip cars, and is transferred to the incline hoist and bin, out of which it is reloaded into the empty tram buckets that have brought green ore to the roast yard. The practice of dividing the bed into roasted material and margins for reroasting, has not worked satisfactorily. Occasionally a bed fails to ignite well, and a large part of it may be entirely unroasted, but it has been found more satisfactory to mix it in with well-roasted ore from other beds and dispose of it in that way, than to try to reroast it. Generally speaking, the amount of wood required has to be increased rapidly as the percentage of rock matter in the ore increases, in order to insure the beds igniting. The roasted ore having been reloaded into tram buckets, is taken to the smelter which is located at the opposite end of the tramway, two miles from the mine.

This method of handling green ore off an aerial tram-line onto a roast yard and back again, is probably the only case in which such a combination is in use. It works quite satisfactorily and is cheaper of construction and operation than a plant arranged for railway. The cost for transportation, including loading at the mine and unloading at the smelter, also including repairs to tramway, is about 12 cents per ton for two miles. When operating at the rate of 150 tons per day the cost for roasting, including labor of building beds and wood, is 15 cents per ton. The reloading onto tramway costs about 30 cents per ton, so that the total cost of transportation and roasting is about 57 cents per ton.

On arrival at the smelter the roasted ore is dumped into a bin out of which it is drawn into charge buggies holding about 1000 pounds each. Two of these buggies are used for each charge.

The roasted ore is smelted with about an equal amount of rocky green ore and 11 per cent. of coke on the charge. Converter slag and cleanings constitute about 25 per cent. of the charge.

The matte produced averages about 14 per cent. Cu and 14 per cent. Ni. This is converted, in silica-lined converters, to a matte containing about 40 per cent. Cu and 40 per cent. Ni and 1 per cent. Fe. The converted matte is crushed and barreled in oak barrels, each barrel holding about 1400 pounds, and is shipped to the Mond Refinery, near Swansea, where the copper and nickel are separated. The nickel is extracted from the roasted matte by means of carbon monoxide, and the copper by sulphuric acid. The smelting and converting of these ores of copper and nickel present special features in detail, but in general the practice is the same as the smelting of copper ores after pile roasting. The furnace matte is kept as low as can be converted in order to reduce the slag losses.

The smelting plant of the Mond Nickel Co. is the same general arrangement as the converter plant of the Anaconda Mining Co., shown in cross-section in Plate XII, Appendix.

The blast-furnaces in the Mond plant are located where the matte melting cupola are in the Anaconda plant.

APPENDIX

Rapid Method for Copper and Nickel Determinations. — Fuse ½ gram ore in a porcelain crucible with about 10 grams $KHSO_4$; run fusion well up on sides of crucible when cooling. Remove from crucible, wash crucible and lid with hot water, dissolve fusion with about 50 cc water and 5 to 10 cc HCL. Filter off SiO_2. Add NH_4OH until slightly alkaline, then make slightly acid with HCL. Precipitate Cu in usual way with H_2S. Filter, wash well with hot water, save filtrate for Ni determination. Any other method for solution may be used, so long as you have solution in above condition for precipitation with H_2S, having little or no nitric acid present.

Copper. — Unfold filter paper and insert about half-way down in a beaker in which precipitation has been made, allowing paper to adhere to side of beaker. Precipitate should now be on lower half of paper. Wash precipitate down into beaker with as little water as possible, add 5 to 10 cc strong HNO_3, and place on hot plate to boil. After boiling for a few minutes wash the paper with a little bromine water till clean, generally one or two applications are sufficient. Wash paper once with hot water and then remove it. Boil solution in beaker very low, say to 1 or 2 cc. Wash down sides of beaker with 15 or 20 cc water, add saturated solution of sodium carbonate until Cu is just precipitated, avoiding an excess. Redissolve this precipitate in a few drops of 50 per cent. acetic acid.

Titration. — The solution should be cold. Add a few crystals of KI (about five or six times the amount of copper present is sufficient) and 5 cc starch solution as indicator; titrate with standard solution of hyposulphite of soda until blue color disappears. The end reaction is very sharp.

Nickel. — Boil the filtrate from copper precipitation until H_2S is expelled. Add about a gram of $KClO_3$ and boil until clear. Remove from plate, wash cover and sides of breaker with water, add 10 cc sodium citrate and cool. Make slightly alkaline with

NH_4OH, using an excess of 1 cc. Solution should smell only slightly of ammonia. Add 5 cc KI solution and 5 cc $AgNO_3$ solution. These will cause a precipitate to appear; titrate with KCN until perfectly clear, let stand two or three minutes, and if no further precipitation appears, the assay is finished. If precipitation does appear, keep adding KCN until a permanent clear is obtained. The end reaction is sharp, one drop clearing solution from what appears a deep turbidity.

SOLUTIONS. — KCN ....45 grams, 98 per cent. in two liters of water,

$Na_2S_2O_3$ ....39.18 grams in 2 liters of water,

KI ....40 grams in one liter of water,

$AgNO_3$..........1 gram in one liter of water,

Sodium Citrate ..213 grams in one liter of water,

Starch ....a lump the size of a pea, dissolved in a few cc of cold water, and added to 4 or 500 cc boiling water boil two or three minutes and use cold.

Bromine water, add hot water to bromine and use.

KCN and $Na_2S_2O_3$ are standardized either with pure metals or ore of known percentage.

A slight deduction will need to be made from the KCN reading for the $AgNO_3$ added.

Slag-Dams

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The Editor:

Sir — In your issue of August 17 there appeared a letter by F. M. Smith, manager of the smelter at East Helena, Montana,

describing a slag-dam which I designed and built in 1898, when superintendent of that plant of the A. S. & R. Company.

As it is the only large dam of its kind of which I have any knowledge, a description of the method of construction will probably be of interest to the readers of the *Press*. Up to the time of its construction the water for the jackets and for general use about the works, was obtained from a ditch taken out of Prickly Pear creek above the plant. As the stream was subject to sudden rises and carried down a large amount of gravel and sand, mud, and general rubbish, the ditch was frequently filled up, and the small dam was often washed away. The jackets and boilers were frequently either filled with mud or short of water, and the general discomfort and interruptions to operations made it desirable that a large storage reservoir for settling be provided.

The creek-bottom at that point is about 1500 feet wide and about 15 feet lower than the general level of the fan-shaped delta of the creek, as it issues from the gorge a mile or more above the works. The delta is a glacial moraine and contains a small amount of placer gold. The slag-dump had been built out from the furnaces so that about one-third of the creek-bottom had been filled up level with the tops of the banks. I proposed to extend a slag embankment like a railway fill across the remaining two-thirds and to put in a spillway so that the flood water could pass, and to use the reservoir formed above the fill for storage. This plan was carried out, and the dam completed about the time I left the East Helena plant to enter the services of the Mond Nickel Company. Mr. Smith's description of the dam is the first mention I have seen of it since that time, the one thing that I have built that will probably outlast the present generation.

The slag from the furnaces was hauled out in twin pots on a revolving trunnion supported on a narrow-gauge railway truck. These pots were of the type built by the Colorado Iron Works and were drawn by a horse. In order to extend the fill in a direct line, the slag had to be poured over its end. To do this it was necessary to have the horse behind the pot when it came near the end of the fill. Standard-gauge rails of about 72 pounds per yard were fitted together with iron ties for the last 60 feet of track, and on the end of one of these rails an iron sheave was fastened. Two other sheaves were placed in a horizontal position so that a ⅜-inch wire rope could pass beneath the rails

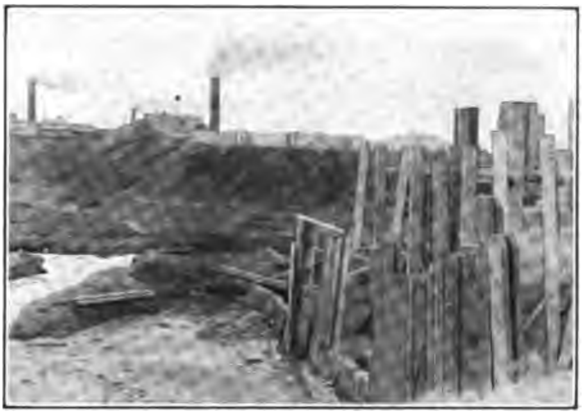


FIG. 35.

FIG. 36.

FIG. 37.

and around the sheaves on the end of the rails extending over the end of the fill. By this means the wire rope was so arranged that when the horse was unhitched from the front of the slag-car, and the end of the rope attached, the horse could be hitched to the other end of the rope and pull the car ahead of him, as shown in the photograph. The photograph is not quite large enough to show the sheave on the end of the rail, but the rope and the rail show as well as the stop-guards on the rail end. The driver rode on the car to operate the foot-brake and led the horse by the bridle. When the fill reached the ends of the rails a short length was put in farther back and the whole frame (with the sheaves attached) moved forward. This process was continued until the fill reached the creek-channel and then a concrete dam on top of sheet-piling was constructed with gates and spill-way to take off the flood-water.

In order to make the bottom of the dam tight, a trench was dug four feet wide and three feet deep in the ground in the center line of the dam before the slag-fill was made. There were also some old ash-dumps near the smelter that had to be cut through and a slag-fill put in place, on account of the porous nature of the ashes. When completed the dam made a lake about 2000 by 1500 feet and 10 feet deep at the lower end. It will probably be used for irrigation purposes long after the smelting industry has faded away for want of ores to smelt, and in the dim and distant future I can imagine some archeologist contending with geologists that it is of human origin and not a lava flow.

COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
JAN. 14, 1893.

PLATE I

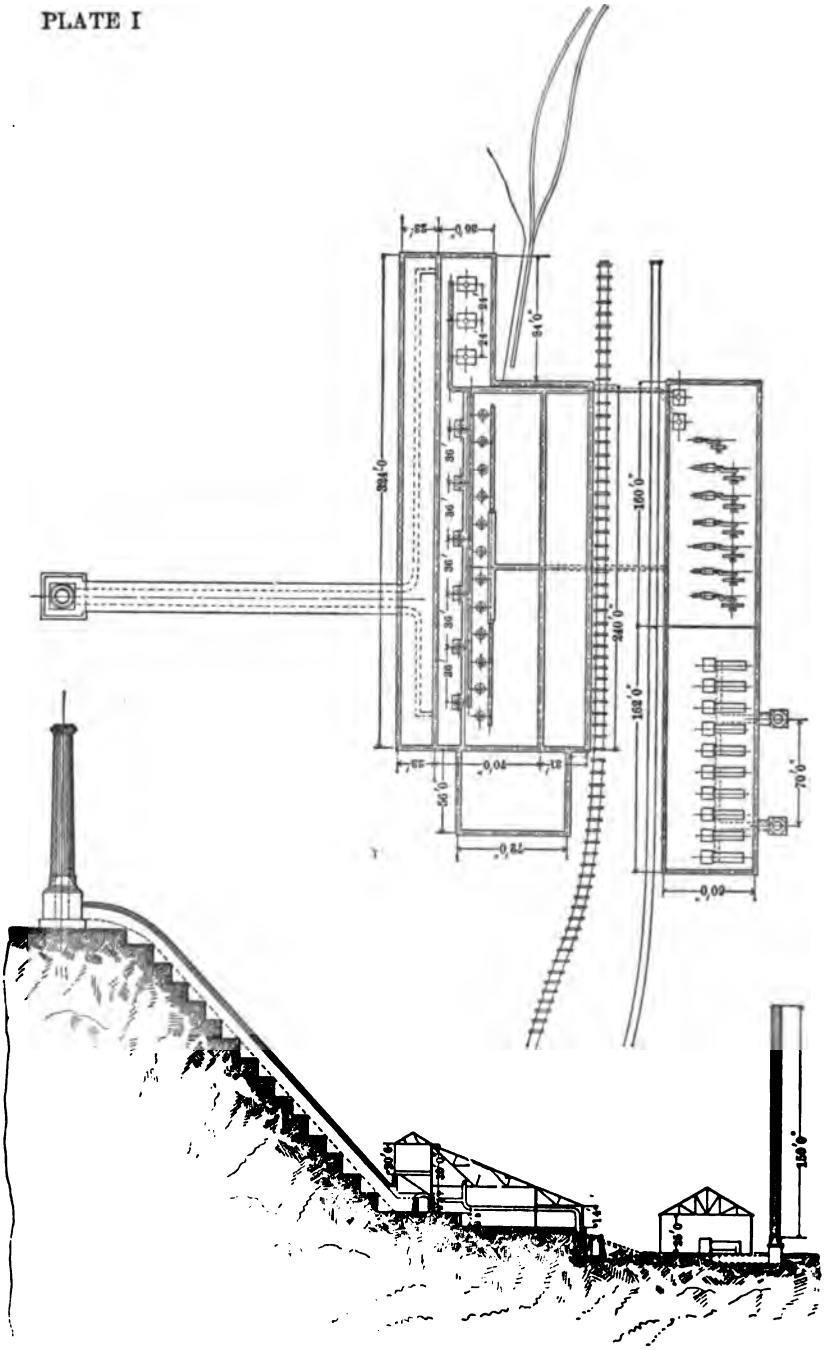


PLATE IV

DETAILS OF CONVERTERS

COPPER CONVERTER PLANT

FOR

ANACONDA MINING CO.

DEC. 4, 1892.

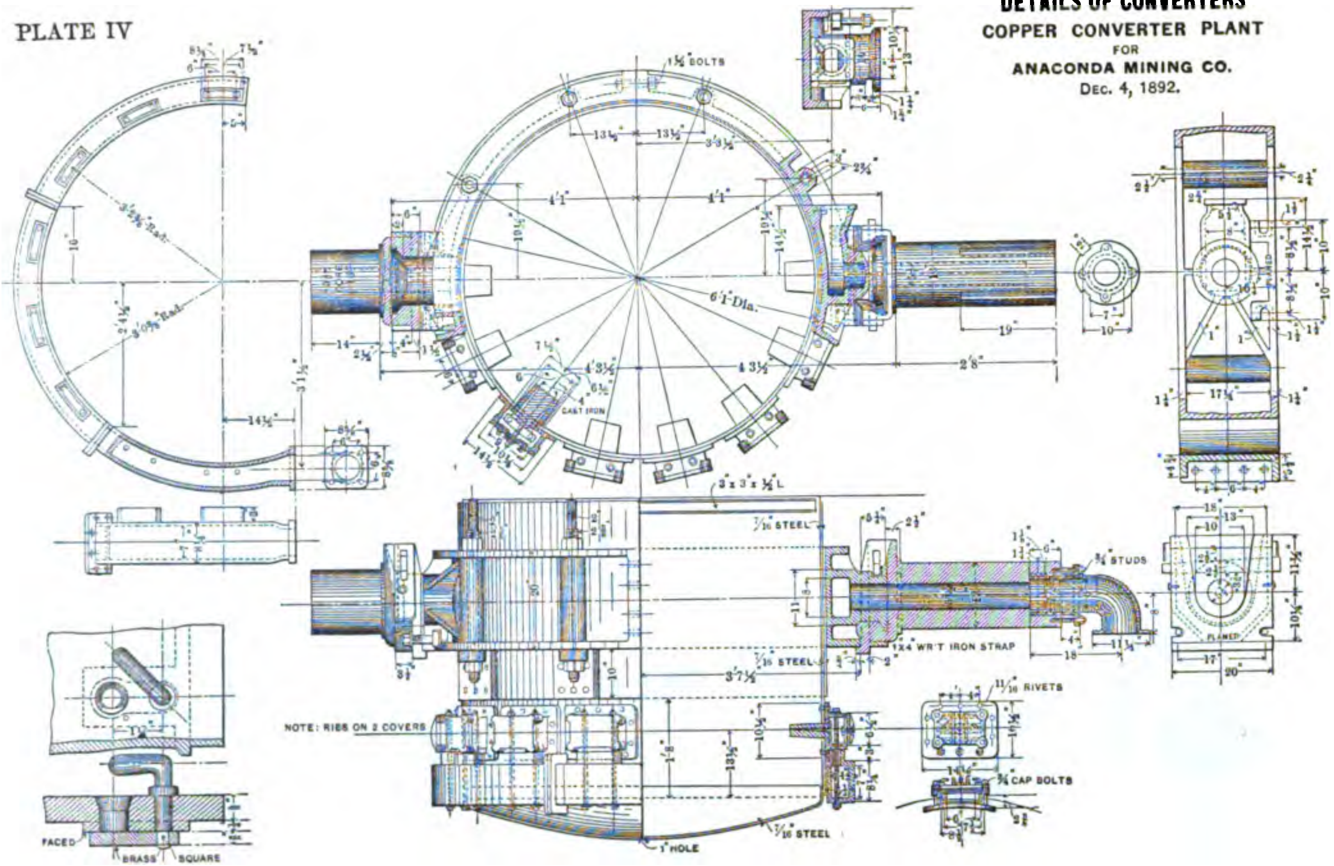
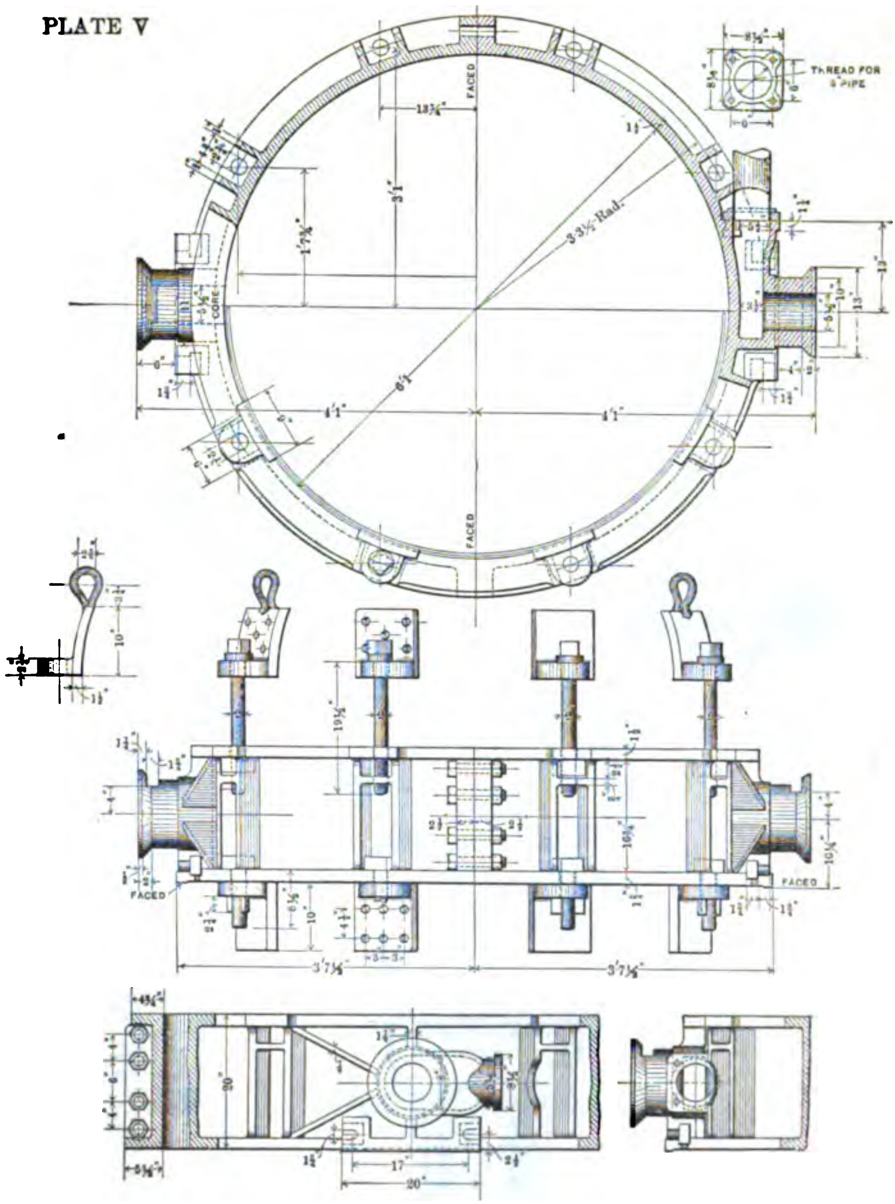
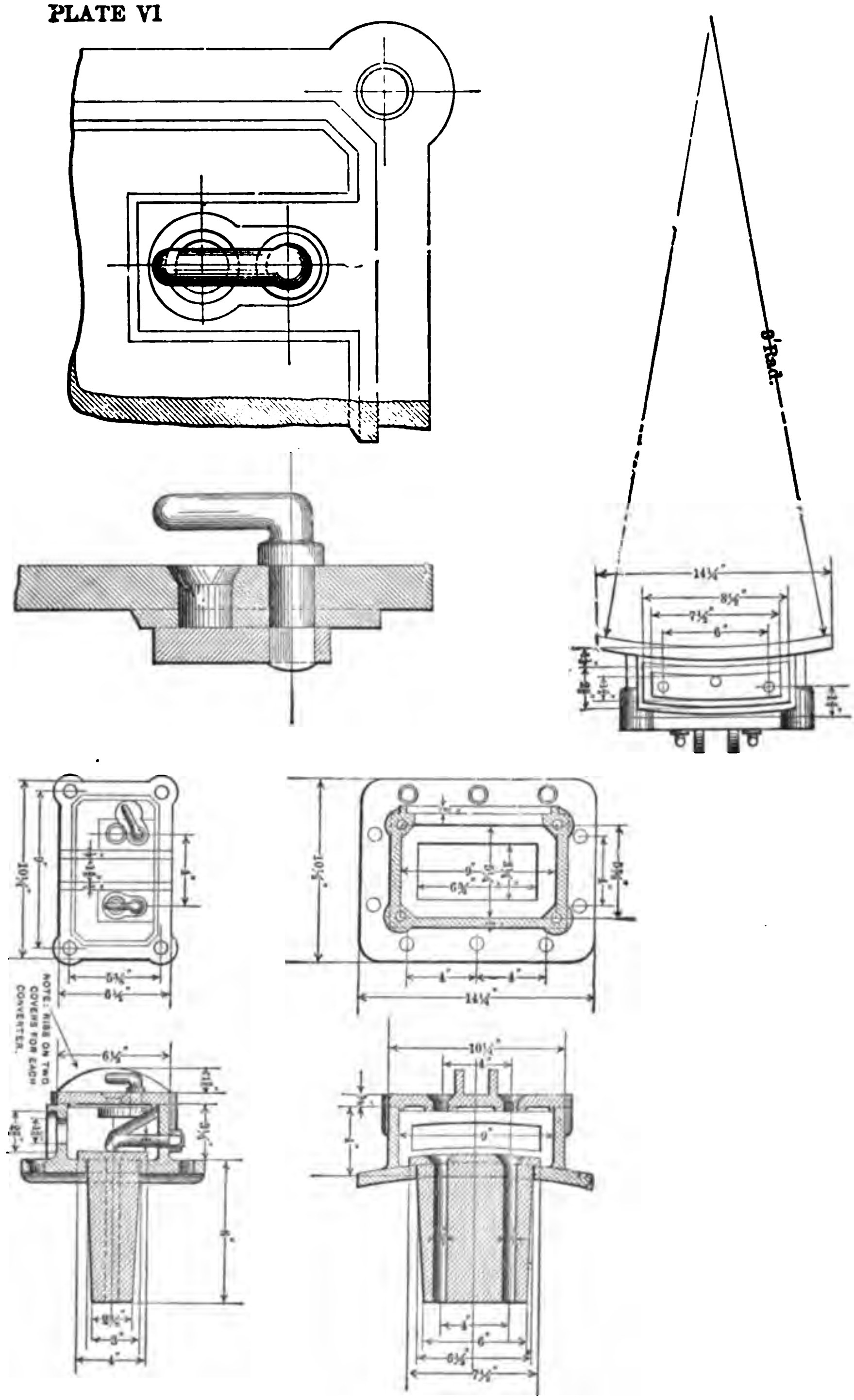


PLATE V



DETAILS OF TRUNION RING ETC. FOR CONVERTERS
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
MAR. 29, 1893.

PLATE VI



DETAILS OF TUYERBOXES AND TUYERS FOR CONVERTERS

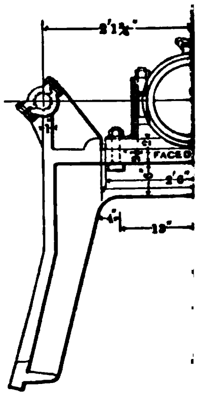
COPPER CONVERTER PLANT

FOR

ANACONDA MINING CO.

FEB. 13, 1893.

PLATE VII



DETA

PLATE VIII

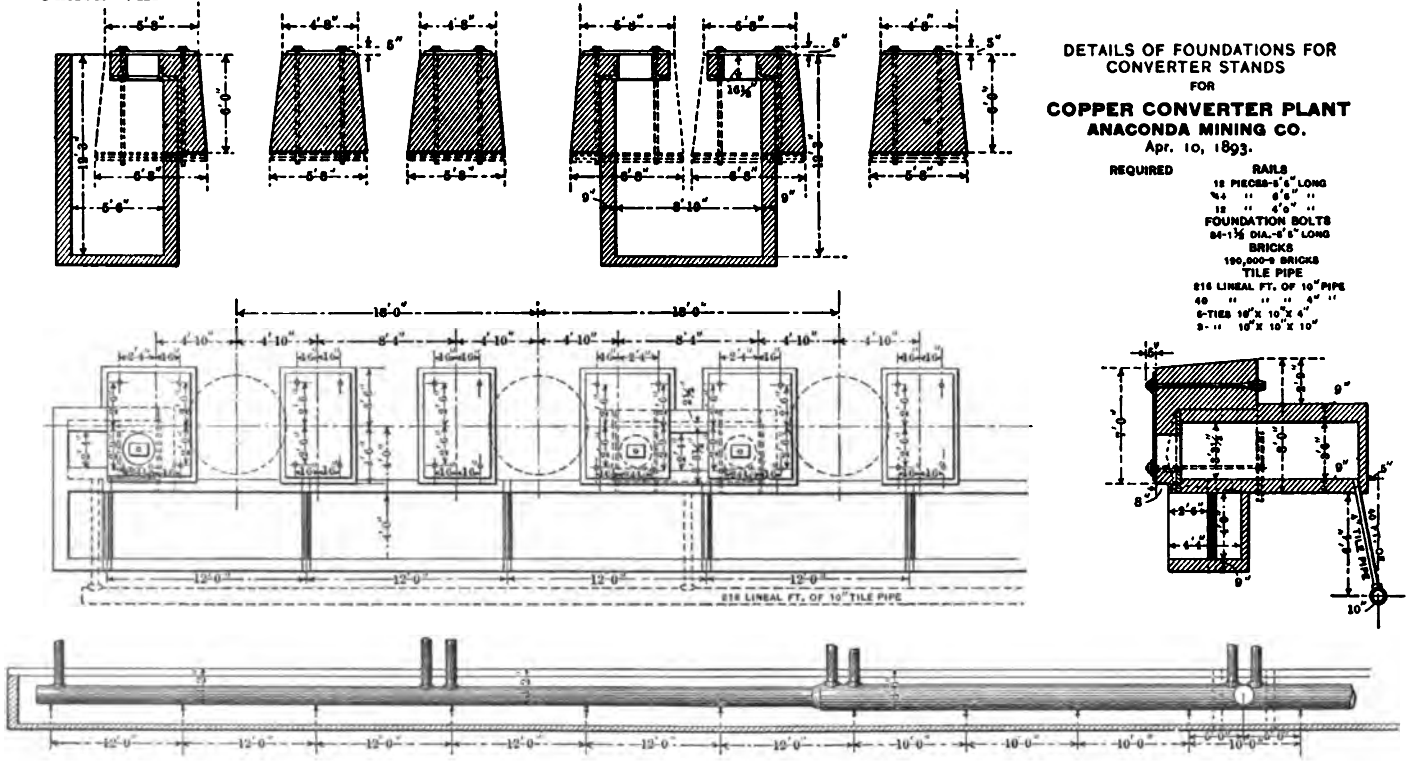

DETAILS OF FOUNDATIONS FOR
CONVERTER STANDS
FOR
COPPER CONVERTER PLANT
ANACONDA MINING CO.
Apr. 10, 1893.
REQUIRED
RAILS
12 PIECES-5'6" LONG
'4 " 6'6" "
12 " 4'0" "
FOUNDATION BOLTS
84-1½ DIA.-6'5" LONG
BRICKS
190,000-9 BRICKS
TILE PIPE
216 LINEAL FT. OF 10" PIPE
40 " " " 4" "
6-TIES 10"X 10"X 4"
3- " 10"X 10"X 10"
216 LINEAL FT. OF 10" TILE PIPE
4" TILE PIPE

PLATE IX

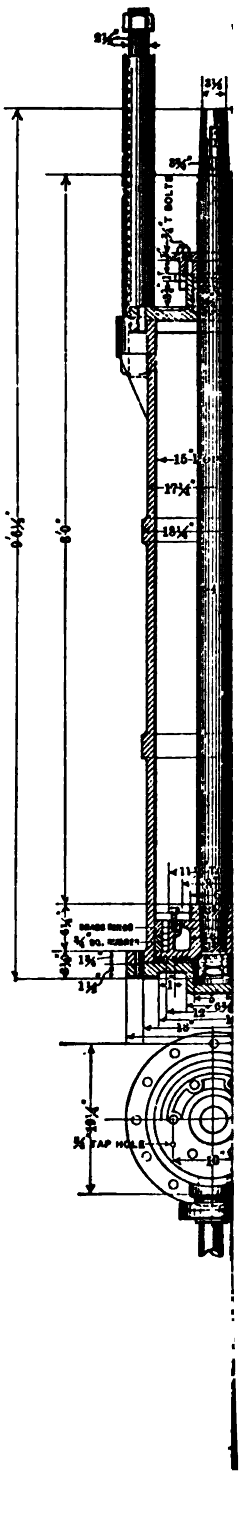
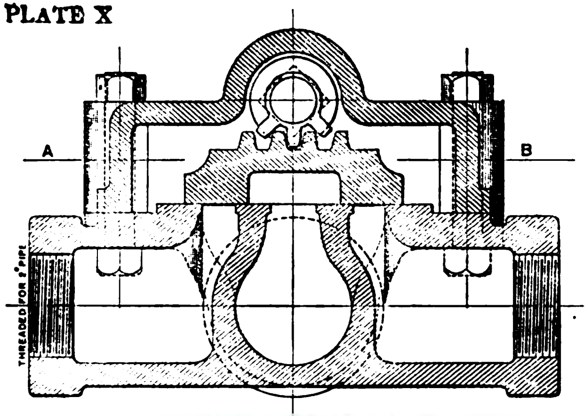
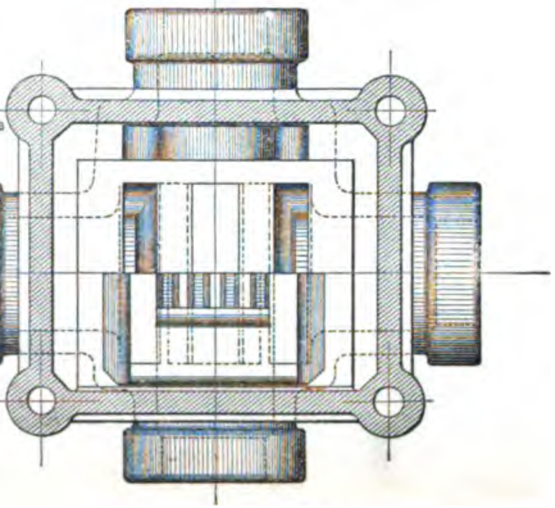
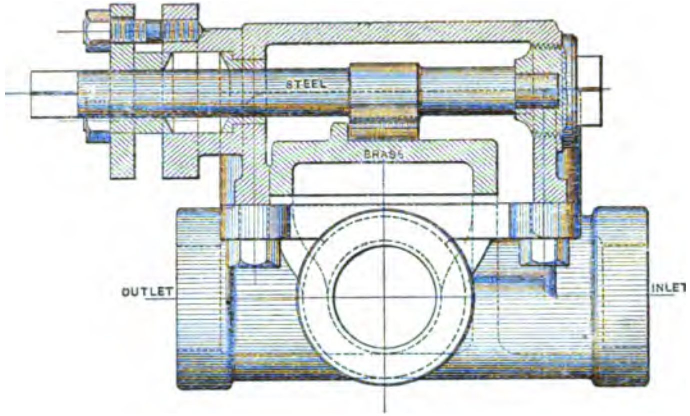


PLATE X



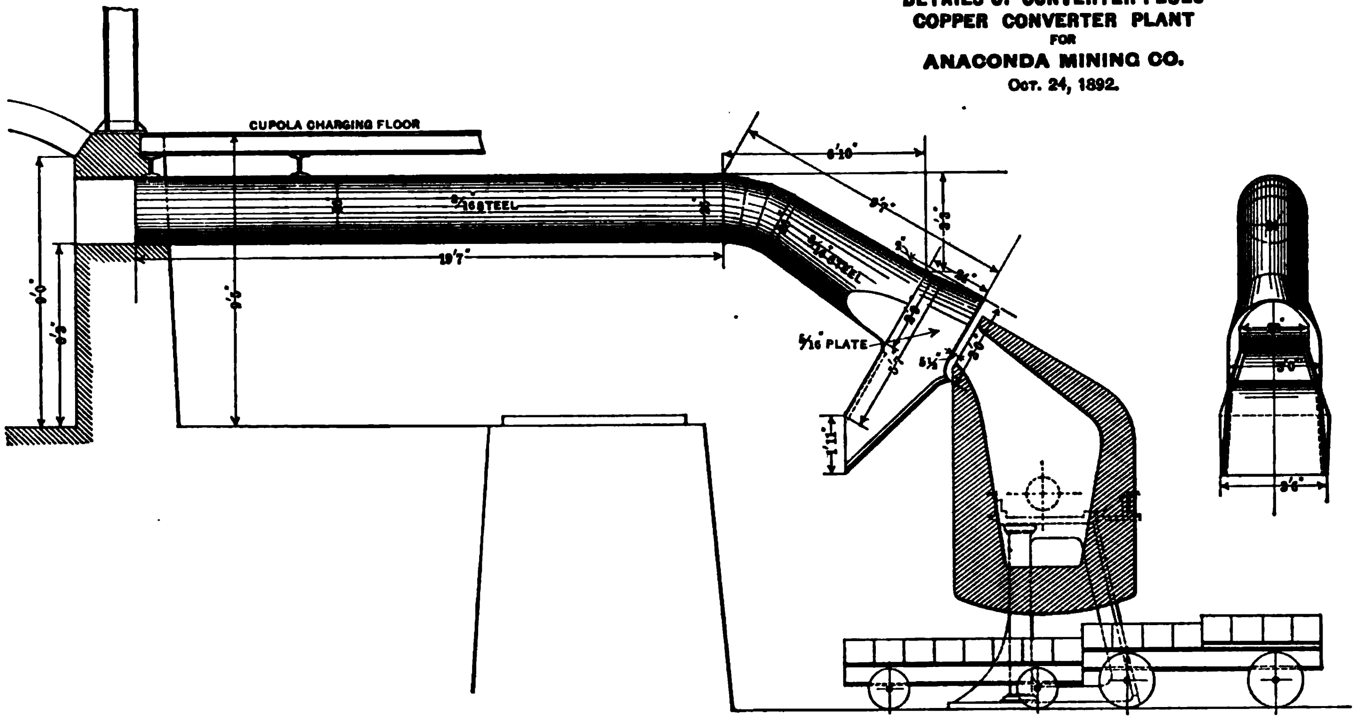
SECTION AT A-B





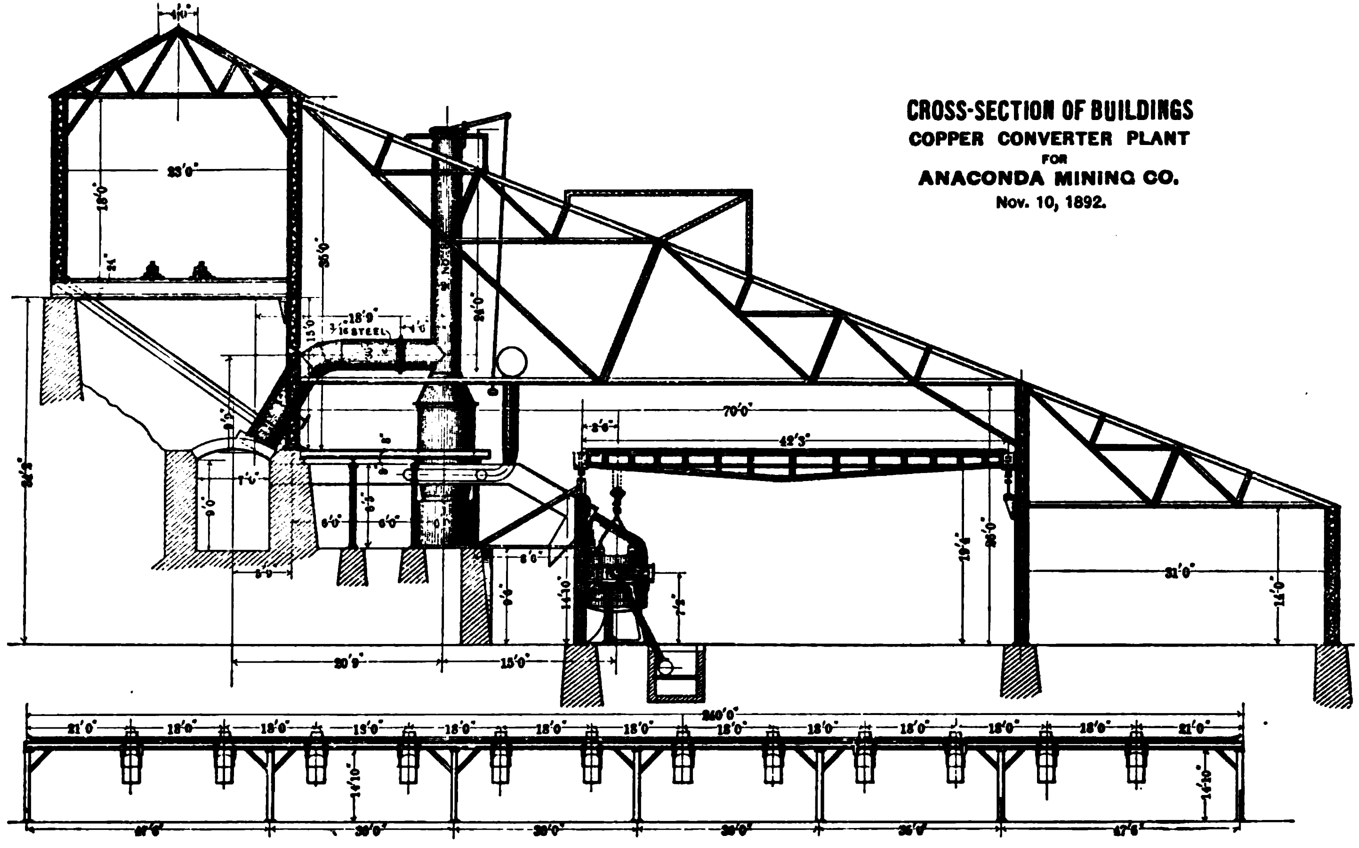
HYDRAULIC FOUR-WAY VALVE
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.

PLATE XII



DETAILS OF CONVERTER FLUES
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
OCT. 24, 1892.

PLATE XIII


CROSS-SECTION OF BUILDINGS
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Nov. 10, 1892.

PLAT

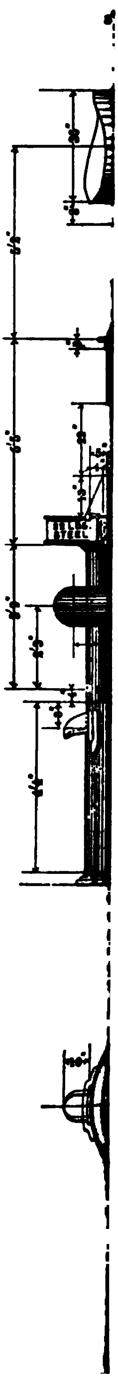
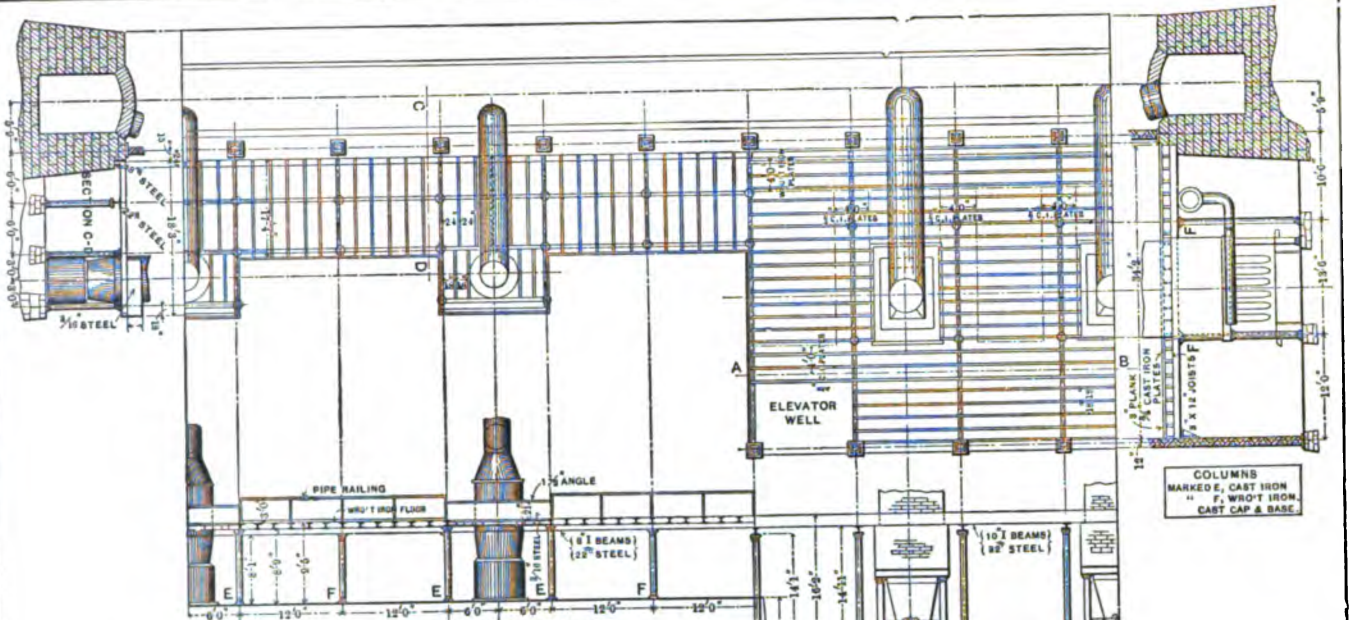
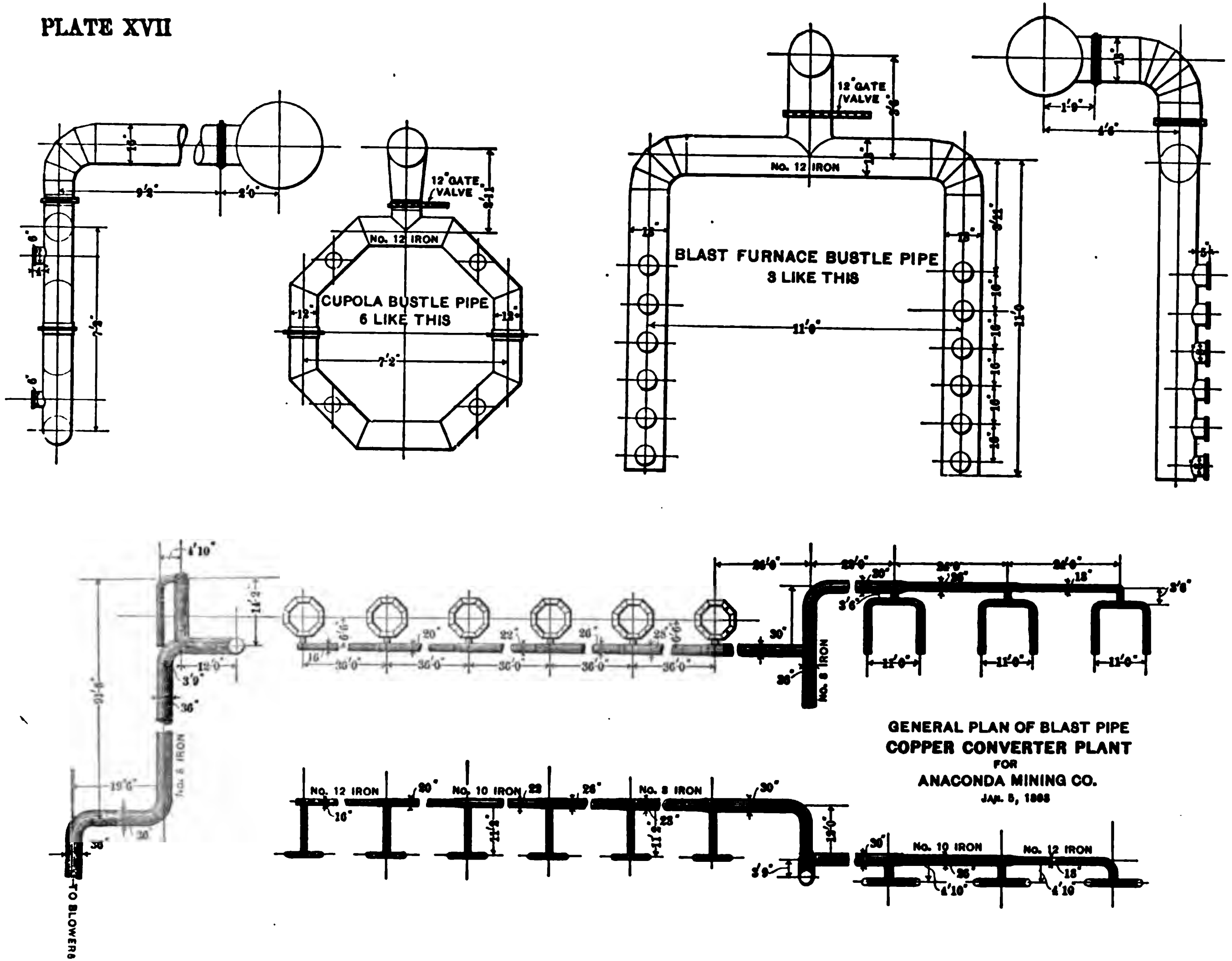


PLATE XV



DETAIL OF CHARGING FLOOR
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.
Nov. 29, 1893.

PLATE XVII



12" GATE VALVE
No. 12 IRON
CUPOLA BUSTLE PIPE
6 LIKE THIS
BLAST FURNACE BUSTLE PIPE
3 LIKE THIS
No. 8 IRON
No. 10 IRON
No. 12 IRON
TO BLOWERS
GENERAL PLAN OF BLAST PIPE
COPPER CONVERTER PLANT
FOR
ANACONDA MINING CO.

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inv'l
us